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UNITED STATES
DEPARTMENT OF THE INTERIOR
HAROLD L. ICKES, SECRETARY

BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE MOUNT ISA MINES, LTD., MOUNT ISA, QUEENSLAND, AUSTRALIA

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BY

J. KRUTTSCHNITT AND V. I. MANN

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TABLE 1. - Cost of glory-hole mining for the first 6 months of 1933

[Ore hoisted, 270,063 tons]

	Labor	Super-vision	Compressed air drills and steel	Power	Explo-sives	Timber	Other supplies	Total
Development:								
In ore	-	-	-	-	-	-	-	-
In rock	-	-	-	-	-	-	-	-
Glory-hole mining	$0.164	$0.012	$0.051	-	$0.064	$0.007	$0.011	$0.309
Transportation (underground)	.124	-	-	$0.043	.010	-	-	.177
General expenses (underground)	.025	.000	-	.009	-	-	.008	.042
Surface expenses[1]	.004	-	-	-	-	-	.006	.010
Total	.317	.012	.051	.052	.074	.007	.025	.538

[1] Includes only surface expense directly applicable to glory-hole mining.

TABLE 2. - Cost of mining by sublevel-stope stoping, center-heading and benching methods, for the 12-month period ended June 30, 1936

[Ore hoisted, 703,924 tons]

	Labor	Super-vision	Compressed air drills and steel	Power	Explo-sives	Timber	Other supplies	Total
Prospecting and developing	$0.056	$0.004	$0.016	$0.001	$0.019	$0.000	$0.001	$0.097
Exploration	.003	-	-	-	-	-	.003	.006
Stope mining	.639	.034	.181	.030	.208	.043	.076	1.211
Transportation (underground)	.171	.007	-	.086	.007	-	-	.271
General expenses (underground)	.106	.006	-	-	.000	.002	.029	.143
Surface expense1/	.020	-	-	-	-	-	.030	.050
Total ore extraction cost	.995	.051	.197	.117	.234	.045	.139	1.778
Capital development	-	-	-	-	-	-	-	.010
Development redemption	-	-	-	-	-	-	-	.090
Total	-	-	-	-	-	-	-	1.878

1/ Includes only surface expense directly chargeable to underground operations.

Table 3 shows the man-hours of labor performed per ton for the various classes of labor in developing and mining 703,924 tons of ore by the sublevel stoping, center heading, and benching methods for the 12-month period ended June 30, 1936.

TABLE 3. - Man-hours labor per ton sublevel stoping, center heading, and benching methods, July 1, 1935, to June 30, 1936

[Ore hoisted 703,924 tons1/]

	Development	Mining	Total
Breaking (drilling and blasting) ...	0.098	0.456	0.554
Mucking	.101	.159	.260
Timbering and filling	.026	.055	.081
Grizzlymen	-	.043	.043
Haulage and hoisting	.041	.214	.255
Supervision	.014	.062	.076
General (underground)	.064	.379	.443
Total underground	.344	1.368	1.712

1/ Excludes shaft sinking.

The following tabulation contains performance based on tons per man shift, division of labor, explosives and timber, and consumption of power and supplies based on tonnage. The tabulation covers the 12-month period ended June 30, 1936, during which 703,924 tons were hoisted.

	Development	Mining	Total
Average tons per man-shift (underground)	-	-	4.66
Average tons per man-shift, total1/.	-	-	3.87
Labor percentage of total cost	5.1	53.7	58.8
Explosives, pounds 40 and 60 percent gelignite per ton	.15	.81	.96
Timber, F.B.M. per ton	.37	1.06	1.43

	Kw.-hr. per ton
Power:	
Air compression	10.98
Haulage and hoisting	3.86
Pumping	4.12
Ventilation	0.56
Total	19.52
Other supplies (in percentage of total supplies and power)	19.0
Supplies and power (percentage of total cost)	41.2

1/ Includes underground shifts and surface shifts properly chargeable to underground operations.

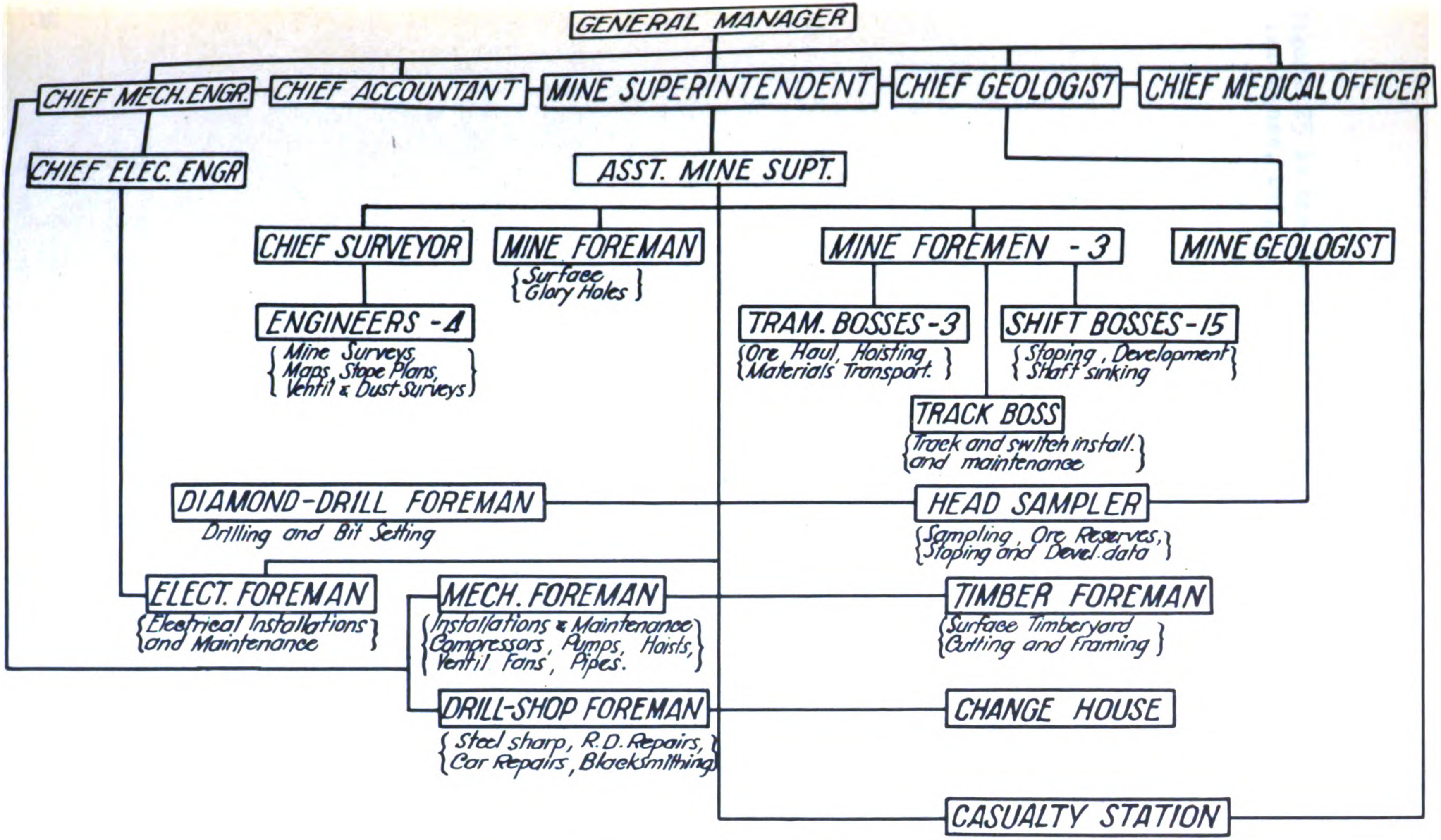
GENERAL MANAGER
CHIEF MECH. ENGR.
CHIEF ACCOUNTANT
MINE SUPERINTENDENT
CHIEF GEOLOGIST
CHIEF MEDICAL OFFICER
CHIEF ELEC. ENGR.
ASST. MINE SUPT.
CHIEF SURVEYOR
MINE FOREMAN
{Surface Glory Holes}
MINE FOREMEN - 3
MINE GEOLOGIST
ENGINEERS - 4
{Mine Surveys, Maps, Stope Plans, Ventil & Dust Surveys}
TRAM. BOSSES - 3
{Ore Haul, Hoisting, Materials Transport.}
SHIFT BOSSES - 15
{Stoping, Development, Shaft sinking}
TRACK BOSS
{Track and switch install. and maintenance}
DIAMOND-DRILL FOREMAN
Drilling and Bit Setting
HEAD SAMPLER
{Sampling, Ore Reserves, Stoping and Devel. data}
ELECT. FOREMAN
{Electrical Installations and Maintenance}
MECH. FOREMAN
{Installations & Maintenance Compressors, Pumps, Hoists, Ventil Fans, Pipes.}
TIMBER FOREMAN
{Surface Timberyard Cutting and Framing}
DRILL-SHOP FOREMAN
{Steel sharp, R.D. Repairs, Car Repairs, Blacksmithing}
CHANGE HOUSE
CASUALTY STATION

Figure 25.—Organization chart, mining department.

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DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

MINING METHODS AND COSTS AT THE MOUNT ISA MINES, LTD., MOUNT ISA, QUEENSLAND, AUSTRALIA[1]

By J. Kruttschnitt[2] and V. I. Mann[3]

CONTENTS

Page

Introduction 3
Acknowledgment 3
History 4
Production 4
Geology 5
Prospecting and exploration 6
  Surface prospecting 6
  Underground prospecting 7
Sampling and ore estimation 8
  Sampling 8
  Ore estimation 9
Development and mining 10
  General 10
  Shafts 12
  Development details 13
    Shaft and winze sinking 13
      Shafts 13
      Winzes 14
    Raises 15
    Drifting and crosscutting 15
  Mining 16
    Mining oxidized ores 16
      Glory holes 16
      Sublevel stopes 17
      Square-set stopes 17
      Flat-back stopes 17
      Methods on the Rio Grande lode 18

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6978."

[2] One of the consulting engineers, U. S. Bureau of Mines, and general manager of Mount Isa Mines, Ltd.

[3] Mine superintendent, Mount Isa Mines, Ltd.

Page

Development and mining - Continued
  Mining - Continued
    Mining sulphide ore 18
      Sublevel stoping by ring drilling 19
      Sublevel stoping by benching 23
  Underground transportation 23
    Haulage-level chute-control 24
    Tramming equipment 24
    Tramming operations 25
  Ore hoisting 25
  Extraction 26
  Explosives 26
  Steel-sharpening and drill-repairing 26
  Drainage and pumping 27
  Ventilation 27
  Fire hazards 28
  Safety methods 28
  First-aid organization and training 29
  Wage, contract, and bonus systems 29
    Contracts 29
    Bonus 30
    Wages 31
  Costs and performance 31
  Organization 35

ILLUSTRATIONS

Figure — Following Page

1. Location sketch 4
2. Plan of diamond-drill prospecting holes, Black Star lode 6
3. Typical diamond-drill-hole section 6
4. Plan of No. 4 main haulage level 10
5. Projected longitudinal section showing main levels 10
6. Man and supply shaft timber sets 10
7. Urquhart shaft timber sets 12
8. Interior and Davidson shaft timber sets 12
9. Details of standard framing for shaft sets 12
10. Main ore-shaft sinking layout 12
11, A, Standard round, main-haulage crosscut; B, standard shaft round, main ore shaft . 12
12. Arrangement for mucking Davidson shaft sinking 14
13, A, Standard round, 7- by 5-foot raise and winze; B, standard round, center headings 14

Figure		Following Page
14.	Hoe-type scraper	14
15.	Scraper slide mucking machine	14
16.	Typical glory hole section	14
17.	Sublevel stope layout, longitudinal type	18
18.	Sublevel stope layout, transverse type	18
19.	Standard stope grizzly	18
20.	Black Star longitudinal sublevel stopes, ring-drilling diagram	20
21.	Black Star transverse sublevel stopes, 24-hole drilling diagram	20
22.	Standard single-type chute	24
23.	Standard double-type chute	24
24.	75-cubic-foot mine car	24
25.	Organization chart, mining department	34

INTRODUCTION

This paper, describing the mining operations at Mount Isa Mines, is one of a series covering mining methods and costs being published by the United States Bureau of Mines.

The Mount Isa property has been in operation since June 1931; during this period operations have changed from the production and treatment of carbonate ores to the mining, milling, and smelting of sulphide ores. This change involved revision of ore-extraction methods.

Stoping methods have been designed to apply to parallel lodes ranging from 10 to over 100 feet in width and dipping at 50° to 60°. The hardness and strength of the ore and enclosing wall rocks, together with the strength and thickness of the country rock between lodes, have permitted the selection of a sublevel stoping method for mining the sulphide ores.

Weights at Mount Isa ordinarily are expressed in long tons of 2,240 pounds, but for comparative purposes all tonnages in this article will appear as tons of 2,000 pounds. All gallon measurements given are Imperial gallons of 277.41 cubic inches. Costs and prices are expressed in dollars converted at the rate of $4 United States currency to the Australian pound.

ACKNOWLEDGMENT

The authors wish gratefully to acknowledge the assistance of R. Blanchard, chief geologist, and of T. C. Gibson, one of the mine engineers, on the illustrations.

HISTORY

Silver-lead ore was discovered at Mount Isa by prospectors Campbell and Miles in March 1923. Thereafter, a mild rush ensued for mineral leases, and soon a small town was established. Early operations that consisted of sinking shallow shafts at favorable places on the outcrops and transporting closely sorted ore to distant smelters were handicapped severely by lack of water, the rigors of an arid tropical country, high cost of trucking 55 miles to the nearest railhead, and inability to ship any but small, high-grade parcels of ore. Individuals and small groups were unable to develop their holdings successfully, and the majority of the mineral leases were acquired by Mount Isa Mines, Ltd., and Mount Isa Silver-Lead Proprietary, Ltd. Late in 1925, Mount Isa Mines, Ltd., absorbed the interests of the other company, bringing the entire field under the management of one company. Development work proceeded in a small way until 1927, when Leslie Urquhart purchased for Russo-Asiatic Consolidated, Ltd., a controlling interest in Mount Isa Mines, Ltd. Diamond-drilling, development, and ore-concentration tests were prosecuted intensively during 1928. A further change in the corporate status of these properties took place about the middle of 1929, when Mining Trust, Ltd., was formed in London and subsequently took over Russo-Asiatic's interests outside of Russia, including Mount Isa Mines, Ltd. Plans were then completed for the erection of mining, milling, and smelting plants. Late in 1930, just prior to completion of the plants, the American Smelting & Refining Co. became interested, through the purchase of a substantial holding in Mining Trust Ltd. Mining, milling, and smelting operations were begun in June 1931, and thereafter have continued without major interruption.

In 1929, a 54-mile branch line was constructed by the Queensland Government Railways from Duchess to Mount Isa, which provided railway connection with Townsville, one of the principal ports of North Queensland. Mount Isa is 603 miles by rail west of Townsville (fig. 1).

PRODUCTION

The following table gives the production, in tons, by classes, from the beginning of mining in 1931 to June 30, 1936.

	Carbonate ore	Sulphide ore	Total
1931	204,961	67,740	272,701
1932	495,071	167,444	662,515
1933	535,740	168,890	704,630
1934	438,357	169,261	607,618
1935	86,159	548,585	634,744
1936[1]	7,199	348,387	355,586
	1,767,487	1,470,307	3,237,794

[1] Includes only the first 6 months to June 30, 1936.

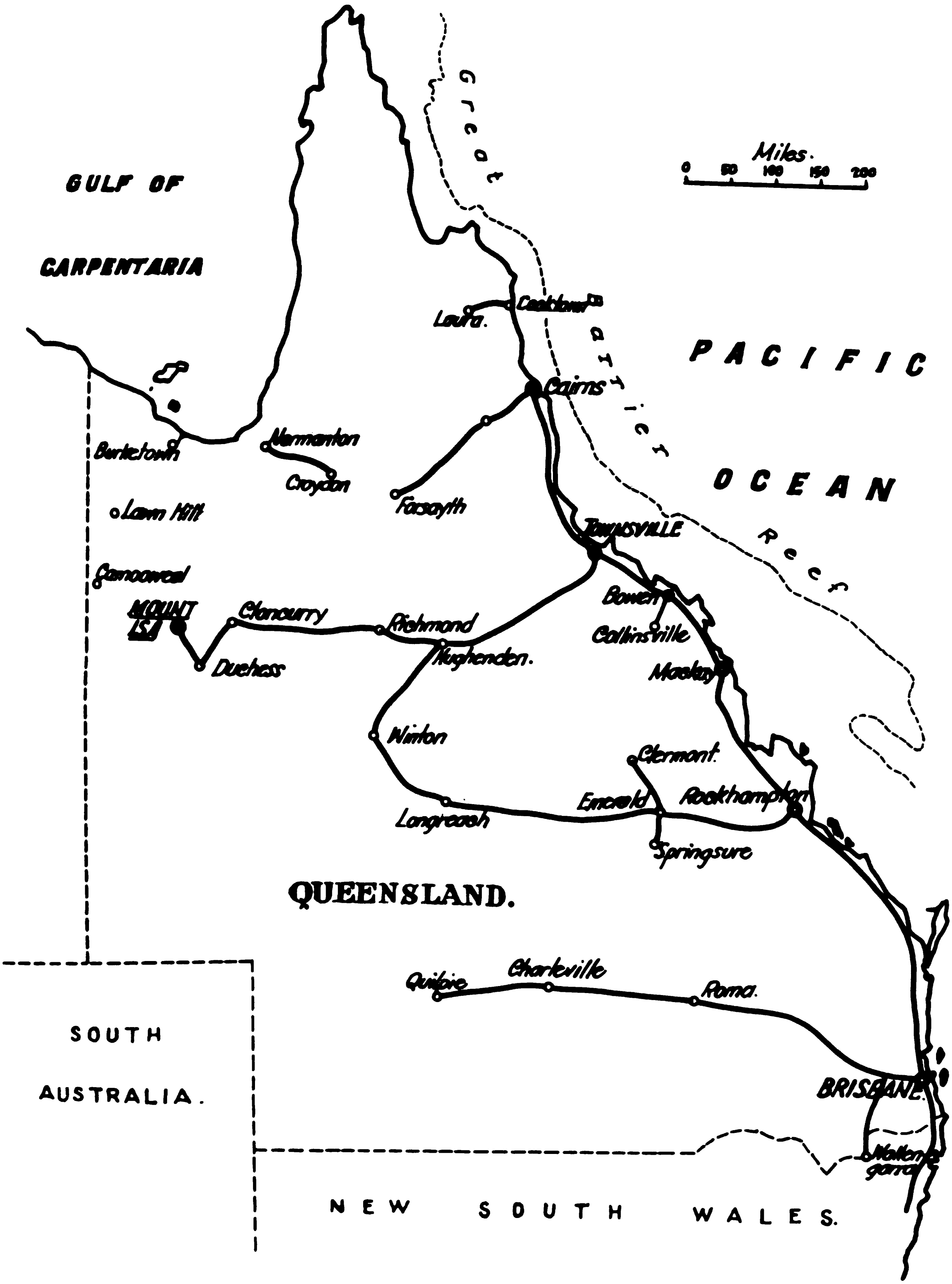


Figure 1.—Location sketch.

GEOLOGY

The ore deposits are replacements in certain strata of the Mount Isa shale series. The series is 2 miles thick and consists of thin, evenly bedded, pre-Cambrian shales lying between quartzites. The shales strike north and south and dip 55° to 60° to the west. Igneous rocks do not outcrop in the vicinity, nor have they been encountered in underground development to a depth of 525 feet or in diamond-drilling to vertical depths of 1,400 feet.

The ore bodies are localized about 1/2 mile from the west edge of the shale series as lenses within zones of shearing and folding. Strike and dip of the shearing conform in general with those of the shale beds. Individual ore lenses vary in size from shoots of a few thousand tons to the Black Star lode with a maximum length of 2,000 feet, maximum width of 210 feet, and a proved depth of 1,200 feet. Lenses are mostly unsymmetrical in outline, with metal content diminishing irregularly along their borders, so that stoping lengths may be from 10 to 50 percent less than the maximum length. The greatest present length of a continuously stoped oreshoot is 1,200 feet, but extreme stoping limits have not been reached.

Mineralogically, the ore comprises silver-lead-zinc sulphides, or their oxidized equivalents, usually interbanded with pyrite and pyrrhotite. The pyrite-pyrrhotite ratio is approximately 7 to 1, and about 7.5 percent of the zinc is replaced isomorphously by iron.

The principal or Black Star lode contains 1- to 3-foot bands of sheared thin shale beds of sufficient number and distribution over a stratigraphic thickness of 160 feet to permit, in part, stoping across the entire width, but in general the shearing and mineralization are most pronounced in a 32-foot hanging-wall panel and a corresponding 40- to 70-foot width in the footwall section. The intervening more massive beds in most places are too low in metal content to warrant stoping. The Black Rock and Rio Grande deposits, which lie stratigraphically 500 to 800 feet out in the footwall of the Black Star deposit are less regular and extensive. The three deposits present, roughly, a formation en echelon. The smaller, higher-grade deposits occur in zones of strong shearing and vigorous mineralization, and although their stratigraphic thickness approaches 100 feet or more, ore bands of 6 to 14 feet alternate with 8 to 25 feet of more massive shale beds almost devoid of mineralization. Consequently, stoping at these places is confined to narrow, high-grade widths. Ore containing less than 10 percent lead is considered low grade, and ore above 10 percent is high grade, irrespective of the other metal content.

In the mineralized area containing the ore deposits, fairly complete oxidation extends to depths of 175 to 200 feet. In this oxidized zone both ore and enclosing wall rocks are weak and in places treacherous, requiring, except in the case of the Black Star glory holes, cut-and-fill or filled square-set systems of stoping. Immediately beneath the completely oxidized ore is a zone 50 feet thick, locally termed the "transition zone."

This zone is characterized by partial alteration, secondary zinc sulphide, and an abundance of magnesium sulphate. The ore in this zone has an adverse effect on the recovery in the flotation mill, and most of this deleterious ore is being left as a level pillar.

Folding and faulting are associated with all ore bodies, but particularly with the smaller, high-grade ones, where their complexity introduces many development and stoping problems. Faults are usually inconspicuous slips, even where vertical displacement amounts to hundreds of feet. Since little variation exists in appearance between the different shale beds, displacement of the faults is often difficult to compute. Search for ore is complicated further by the fact that all folding and faulting observed to date are pre-ore, and mineralization usually weakens or dies out completely within 5 to 30 feet of them, necessitating development work in low-grade or barren rock. The chief concern caused by folding and faulting, however, is the accompanying weak hanging wall. In oxidized ground this has called for filled square-set stoping practice, and in both oxidized and upper sulphide ground frequent hanging wall support by headboards or filling, or both, is demanded.

The Black Star ore body, which furnishes 90 percent of the mill feed, is low grade. The average metal content for the whole is 4.83 ounces of silver per ton, 8.17 percent lead, and 8.07 percent zinc. The lead content of the Black Rock and Rio Grande lodes is two to three times, the silver content over three times, and the zinc content usually one-third to one half that of the Black Star lode.

Commercial ore extends to the surface in the principal ore bodies now being worked, but at least one major ore body has been disclosed by diamond-drilling whose outcrop consists only of leached pyritized shale and in which ore does not come within 800 feet, vertically, of the surface.

Outcrops of the oreshoots are inconspicuous. They consist of irregular patches of oxidized lead minerals and limonite, which project locally through the alluvium. They occur mostly along hanging-wall or footwall slopes of extensive, prominent, siliceous masses that are themselves barren and whose talus at their base tends to obscure the lode outcrops. The siliceous masses rarely cause stoping problems; but where such masses are absent, the softer, oxidized shale hanging wall of the oreshoots locally develops weakness and requires support to avoid collapse near the surface.

PROSPECTING AND EXPLORATION

Surface prospecting

Original surface prospecting was comparatively simple, due to the small amount of overburden and the continuity of ore-bearing shales. The oxidized ore outcrops were exposed by trenching throughout the length of the mineralized lenses, which made possible the planning of a comprehensive diamond-drill prospecting program to test the continuation of ore at depth in the large low-grade Black Star lode.

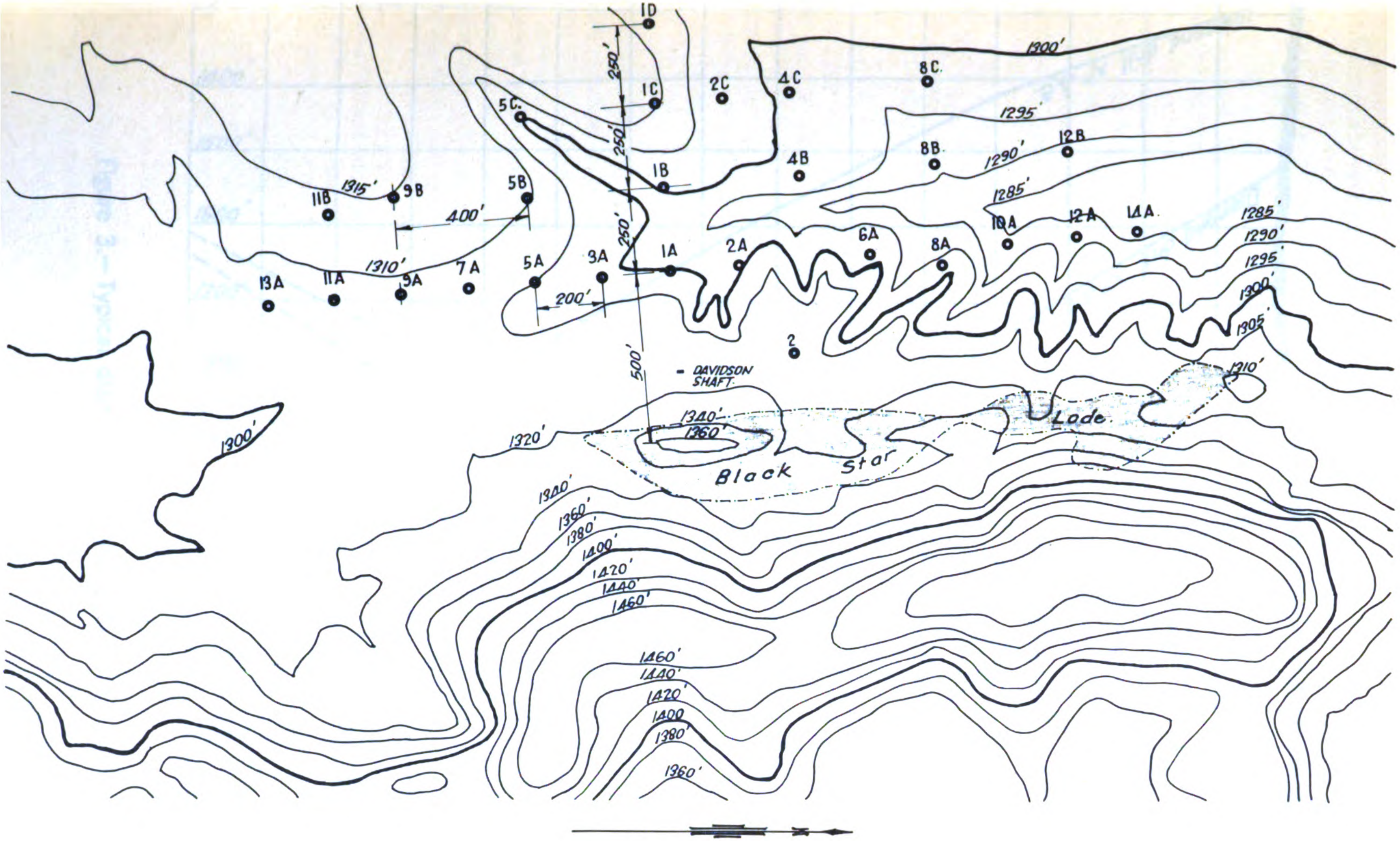


Figure 2.— Plan of diamond-drill prospecting holes, Black Star lode.

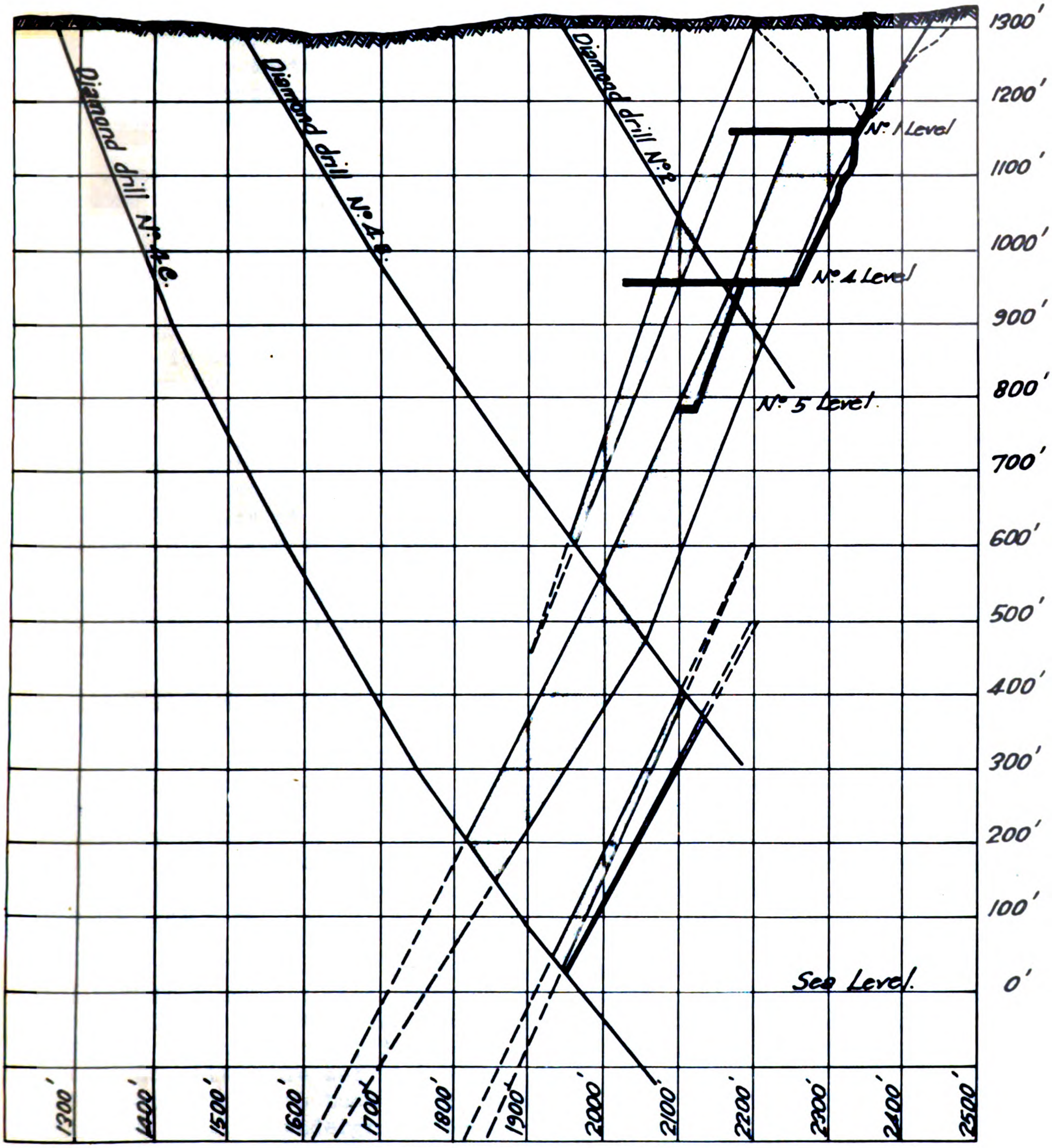


Figure 3.— Typical diamond-drill-hole section.

The narrower high-grade lenses of the Black Rock lode, being overlain by considerable thickness of leached and brecciated siliceous shale, made drilling inclined holes from the hanging-wall side to intersect the lode at depth extremely difficult and costly. Inclined holes were drilled successfully in testing the Rio Grande lode, but only a few holes were drilled, and these gave limited information of value for planning subsequent development.

A considerable amount of prospecting was done through small shafts put down in ore to a depth of approximately 85 feet and by driving along the lode and crosscutting at various intervals. The workings and assay records remaining from this type of prospecting provided useful data for calculating grade and tonnage of oxidized ore.

A plan of the layout for diamond-drill prospecting-holes in the Black Star lode is shown by figure 2, with a typical section through a line of holes shown by figure 3. A base line running roughly parallel to the strike of the outcrop was laid out 430 feet west of the lode hanging wall. Along that line, at 200-foot intervals, the "A" line of holes was drilled. At 250-foot distances, respectively, from the base line and parallel to it, "B" and "C" lines were then laid out with holes drilled at 400-foot intervals. A "D" line of holes was planned, but only one hole in this line was drilled. All holes were planned to be drilled normal to the average dip, thus giving information at regular intervals of depth. Variations due to hole deflection and change of dip in the formation have since been noted, both from hole surveys and where holes have been intersected by mine headings.

Drill cores, properly numbered, were filed in covered wooden trays, visual examination made, and the description and percentage of core recovery tabulated. Mineralized sections of core were split by hand, one-half assayed and one-half kept in proper position in the tray. Rejects from the half taken for assay were used for metallurgical tests in small laboratory machines of the mill pilot plant. These tests served as a guide and basis for establishing the concentration flow sheet, particularly for sulphide ore treatment, as no other samples of primary ore were obtainable at that time.

Surface diamond-drill equipment consisted of type C Sullivan drills of 1,500-foot capacity and operated by steam from small, portable, wood-fired boilers; core diameter was 1-1/8 inches. Each plant had a water-storage tank and a small, duplex feed pump for serving both boiler and drill. The bits were set with eight black carbon stones per bit. At present the carbons have been partly replaced by bort in the bits, particularly for drilling underground prospecting holes.

Underground prospecting

In the oxidized zone, prospecting was done mostly by drifting along the lode and crosscutting at intervals to ore limits. All such work was accompanied by close geologic mapping, so that work in barren shale was kept at a minimum. Faulted, folded, or other irregular sections of the lode were recorded carefully, and their probable effect on ore continuity and stoping was appraised. These drifts usually were on or near the lode footwall, from which raises and chutes were run preparatory to stoping. Raises or winzes between levels completed prospecting in the oxidized zone.

Now that practically all stoping is in the sulphide zone, the method of prospecting and delimiting ore has been modified. In the Black Star lode, the main drifts are located either in the footwall or hanging wall portion; the plans are made from data from surface drill holes. Short, horizontal holes 50 to 200 feet long are drilled at 200-foot intervals from the main drift to lode limits. Limits are determined from inspection and assays of mineralized core sections. A small, compressed-air-operated, prospecting type of drill with a 400-foot capacity and using size "E" rods has proved most satisfactory for this work. These short drill holes also provide data for laying out a more orderly system of level development.

Recent work in the Rio Grande lode has revealed many irregular faults that cut up and destroy continuity of the narrow high-grade lenses, making it difficult to follow ore with prospecting drifts. In this section, underground prospecting drill-hole data has been of much assistance in locating lode variations, thus allowing level and stope preparation work to be planned economically.

Diamond-drill cores are examined by the geologist or head sampler and mineralized sections marked for assay. These sections are split by hand; one-half is taken for assay purposes and the other half is kept in the core box in proper position. Results of core assays are plotted on assay plans or drill-hole sections.

SAMPLING AND ORE ESTIMATION

Sampling

A sampling crew of four to six men, supervised by a head sampler, is employed in the mine. Samples are brought to the surface daily and sent to the assay office for determination of silver, lead, zinc, and, when required, iron and sulphur. Records are kept of all data pertaining to each sample; assays are plotted currently on mine assay maps.

Drifts following ore are sampled by cutting a groove across all mineralized beds exposed. Each groove is taken as one sample. The sample along drifts in the Black Star ore body is taken at 15-foot intervals. In the high-grade lenses of the Black Rock and Rio Grande ore bodies the spacing is 10 feet. Check samples, when required, are taken at the location of the original sample.

Crosscuts in ore are sampled by cutting continuous horizontal grooves along both sides. Corresponding samples from each side over a 5-foot width, are combined and taken as one sample. Where large folds in the beds, or faults, occur within limits of the opening, the lines for sample grooves are marked carefully by the head sampler or mine geologist to prevent sampling duplicate beds. In such instances, departure may be made from the standard 5-foot width. Sampling procedure for crosscuts in all parts of the mine is the same.

Inclined raises and winzes are sampled at intervals of 5 feet down the dip, with grooves cut normal to the beds on each side of the opening. These samples are assayed separately. Vertical raises or winzes are sampled at the same interval, with grooves cut horizontally.

Sampling of all development openings progresses at some distance behind the working face so as not to interrupt operations. Where headings approach ore limits, or lean sections are encountered in the lode, special face samples are taken to determine whether material broken is ore or waste.

Sublevel stope workings are sampled the same as drives or crosscuts, transversely or longitudinally, depending on the stope layout. In transverse stopes of the Black Star section, only the sublevel midway between main levels has been sampled regularly. In longitudinal stopes of this section, present practice is to sample crosscuts driven for pillar cut-offs on each sublevel, to supply additional data for ore estimation. In the Rio Grande sublevel stopes sampling is done only along the undercut and alternate sublevels.

Broken ore samples are obtained one day each week at the ore-pocket grizzly. A grab sample is taken from the top of all cars dumped, the sample from each stope being kept separate and a record made of the number of cars represented by each sample. The grade determined by these samples is termed the stope extraction grade. It serves as a check against computed grade for each stope.

Final grade for run-of-mine ore is based on a sample taken as the ore passes from the mill crushing plant to the storage bins. Part of this is assayed daily; the balance is retained and assayed as a composite sample for the month, thus serving as a check on the average of the daily samples. Samples for determination of moisture content of the ore are taken from the same position in the mill at regular intervals throughout the day. Drill-hole sludge samples are taken only when the drill penetrates mineralized zones where core recovery is poor.

Ore estimation

Ore reserves are adjusted semiannually to correspond with development and extraction. The grade is adjusted when additional data show such alterations to be justified. Reserves are divided into two classifications - "developed ore" and "prospective ore."

Developed ore includes all ore blocked out by development or by extensive diamond-drilling. The tonnage in developed sections is calculated from engineers' measurements by means of a factor of 11 cubic feet per ton for ore in place. All available information is considered in making calculations, with necessary allowances or deductions conformable with stoping methods. Calculations of tonnage outlined by extensive diamond-drilling are made from sections through each hole or line of holes. Ore widths are plotted from core assays, resulting areas are planimetered, and volumes between sections are calculated. Where ore is not continuous, laterally, in two or more holes, no allowance is made for extension beyond limits of the last continuous section. Broken ore in stopes is kept at a minimum and is included in reserves with developed ore.

Prospective ore comprises ore left as pillars where extraction must be long delayed and ore outlined by incomplete development and by limited diamond-drilling.

Grade of ore reserves is determined from available assay records for each section or group of sections. The known geological and mining factors that may affect the extraction of ore are given consideration in mathematical calculations for the grade of each section. The large Black Star ore reserves below the horizon of immediate mining operations have been calculated on grade of drill core assays alone. Grades thus calculated are subject to revision as each new level is developed, giving more accurate data for the immediate horizon. In low-grade sections a deduction of 5 percent is made in arriving at the final estimated grade to allow for dilution and for possible lean portions in the block not revealed by sampling. In narrow, high-grade lenses a deduction of 15 percent is made to offset dilution from stope development and in mining poorly mineralized sections adjacent to faults or other irregularities commonly found to exist.

Mine plans and sections are drawn to a scale of 50 feet to 1 inch. Assay and geologic maps are drawn to the same scale, so that information may be readily transferable without adjustments.

Mine models of horizontal glass plates have been constructed for the Rio Grande and Black Rock lodes and are kept up currently by the geological department. A model for the Black Star section is being constructed with plates representing vertical cross sections at 100-foot intervals. The models are available to the mine operating staff and are of much use and assistance in planning development work.

DEVELOPMENT AND MINING

General

The mine has been developed down to the present main no. 4 haulage level through four vertical shafts. They consist of two large or main shafts and two smaller, auxiliary ones. Location of the shafts and their relative positions are shown by figure 4.

The smaller Davidson and Lawlor shafts, sunk during the early development period, extended approximately to the bottom of the zone of oxidation. Subsequently they were extended to the first main, no. 4, haulage level and

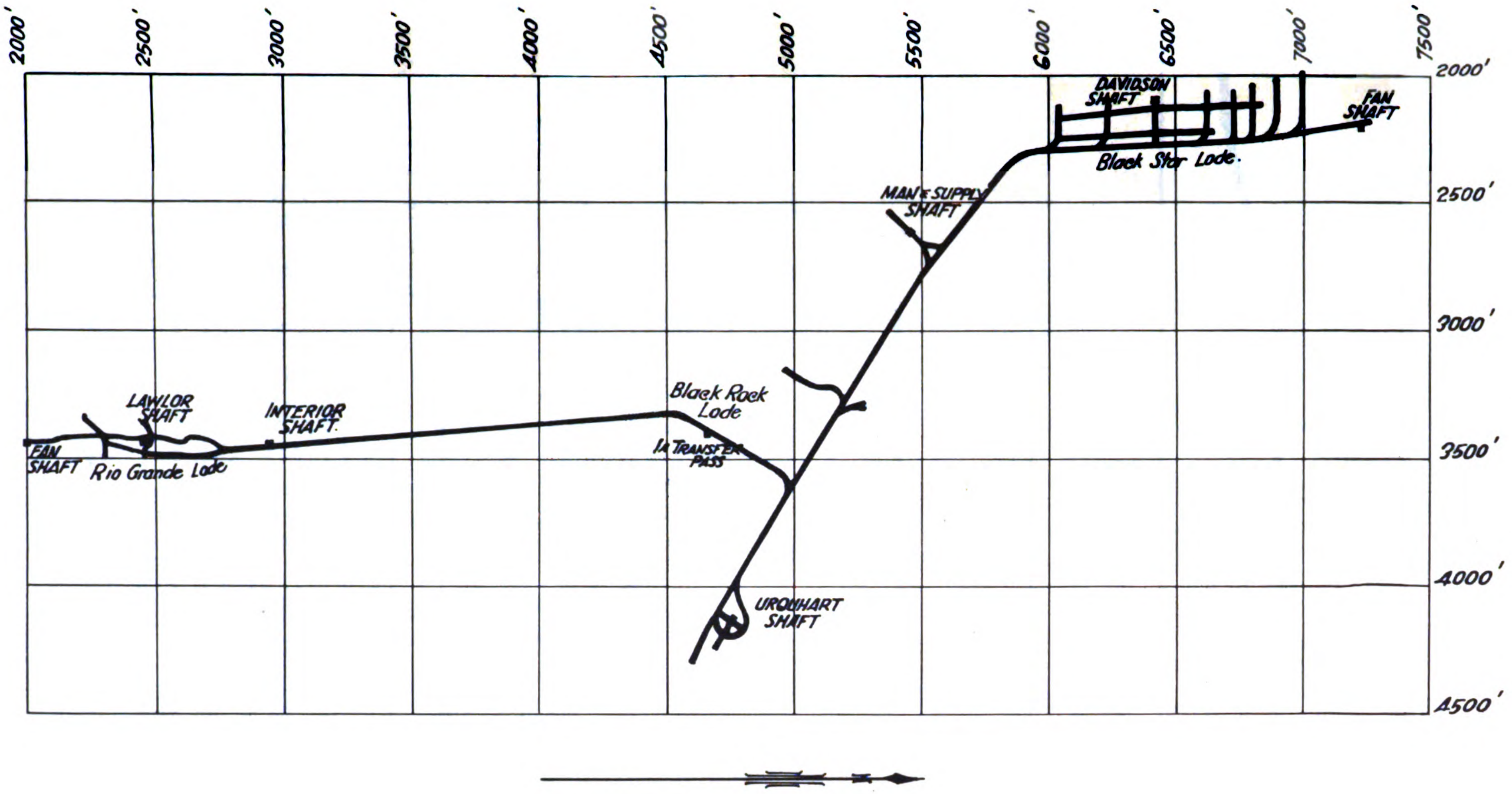


Figure 4.— Plan of no. 4 main haulage level.

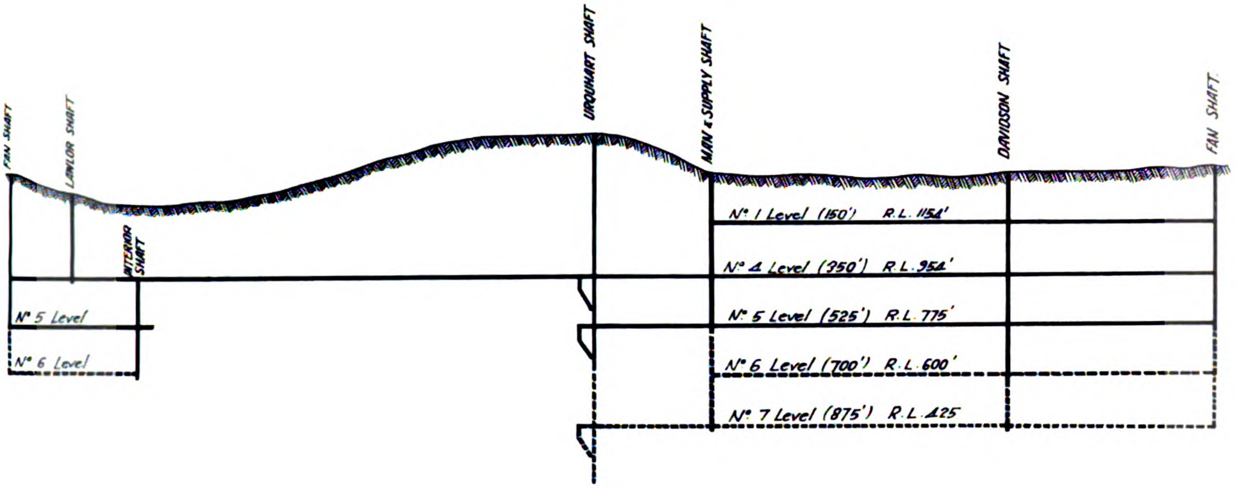

FAN SHAFT
LAWLOR SHAFT
INTERIOR SHAFT
URQUHART SHAFT
MAN & SUPPLY SHAFT
DAVIDSON SHAFT
FAN SHAFT
Nº 1 Level (150') R.L. 1154'
Nº 4 Level (350') R.L. 954'
Nº 5 Level (525') R.L. 775'
Nº 6 Level (700') R.L. 600'
Nº 7 Level (875') R.L. 425
Nº 5 Level
Nº 6 Level

Figure 5.—Projected longitudinal section showing main levels.

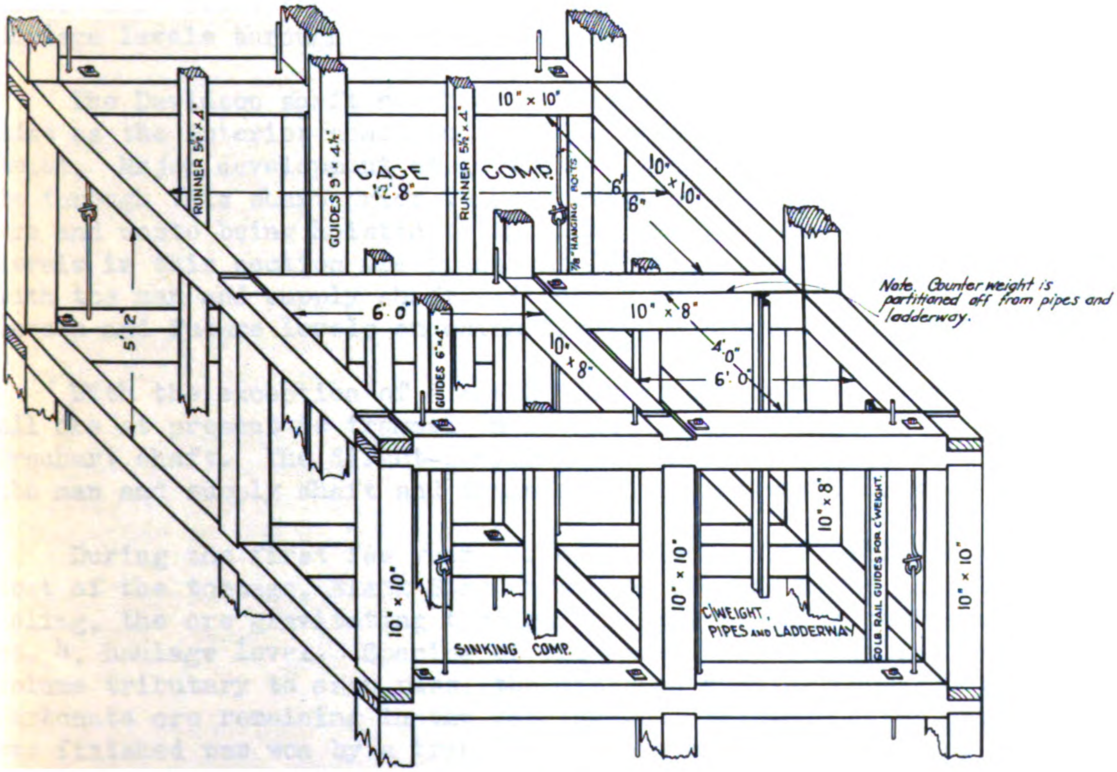


Figure 6.—Man and supply-shaft timber sets.

utilized for general stope development in the lodes they penetrate. The main haulage level was partly driven from headings off these shafts.

The two larger shafts, man and supply and the main ore (Urquhart), were planned after the ore bodies had been delineated for the most part, and were located away from the ore bodies, with due respect to plan of underground workings and position of surface plants.

The no. 4 haulage level (fig. 4) is 9 by 8 feet in cross section and at an elevation of 370 feet below the collar of the man and supply shaft. The second, no. 5, haulage level is 175 feet below no. 4 level. Haulage levels below no. 5 are planned at intervals of 350 feet.

At present, the ore expected at the Rio Grande below no. 4 level does not justify extension of the long haulage drifts to that ore body. Partly because of this fact, and partly to facilitate ore extraction, the Lawlor shaft was abandoned below no. 4 level and an interior shaft sunk from a nearby position. Development and ore extraction in the Rio Grande lode below no. 4 level are now carried on from this shaft; the level interval is 175 feet. The interior shaft is equipped with a slow-speed, heavy-duty hoist, with a cage large enough to permit hoisting 75-cubic-foot cars. The ore is hoisted in cars and transported along the haulage level to the Urquhart shaft. Subsequent work will continue by this method unless sufficient tonnage is developed to warrant extension of succeeding haulage levels through to the Rio Grande section.

The Davidson shaft has been enlarged below no. 5 level to the same size as the interior shaft and will be equipped with a duplicate-type hoist. Major development of lower levels of the Black Star ore body will be through this shaft prior to completion of the next lower haulage level, ore and waste being hoisted to the haulage level above for disposal. Mining levels in this section are spaced at 175-foot intervals and are connected with the man and supply shaft. Figure 5 shows the layout of the present levels and future levels on which development work is in progress.

With the exception of a small current tonnage for direct smelting, all ore at present is trammed on the main, no. 4, haulage level to the Urquhart shaft. The direct-smelting carbonate ore is hoisted in cars at the man and supply shaft and transferred in railway trucks to the smelter.

During the first few years of operation, when carbonate ore constituted most of the tonnage, Black Star carbonate ore was mined largely by glory-holing, the ore gravitating through passes to loading chutes on the main, no. 4, haulage level. Spacing of the passes was controlled by the area or volume tributary to each pass, the distance varying from 140 to 150 feet. Carbonate ore remaining in the hanging-wall triangle after surface mining was finished was won by a transverse stope and pillar method. The ore was transferred upon no. 1 level by both hand-tramming and scrapers to the transfer passes previously utilized for passing glory-hole ore to the haulage level.

Rio Grande and Black Rock carbonate ore was mined by shrinkage stoping where strength of hanging wall permitted. In the weaker ground, both incline and flatback cut-and-fill and filled square-set stoping methods were used. The ore was transferred at bottom of the oxidized zone by handtramming, scrapers, or both, to a central ore pass at each mine that extended to the main, no. 4, haulage level below.

Sulphide ore above no. 4 level, at all mines, has been stoped into chutes on that level and loaded directly into the 75-cubic foot cars.

Shafts

The man and supply shaft has three compartments; details are shown in figure 6. A single large cage, 12-1/6 by 5-1/2 feet, with counterbalance, is operated by a Fraser & Chalmers (England) single-drum, double-reduction geared-type hoist with a Ward-Leonard control system. The hoist motor is rated at 220-horsepower at 650 r.p.m. Light loads are hoisted at 550 feet per minute; by means of two-gear ratios incorporated in the design, heavy loads may be hoisted at a speed of 150 feet per minute. Brakes are air-operated, post type, dead-weighted and designed so that pressure is required for their release. In the event of air failure the brakes are applied automatically by gravity and current supply is cut off.

Underground employees, steel, timber, tools, explosives, and other store are handled through this shaft. When not under manual control, the hoist may be operated automatically by push-button operation from inside the cage.

The cage weighs approximately 8.8 tons, being counterbalanced by an 18,000-pound cast-iron weight running in 60-pound-per-yard steel guide-rails. The cage has a capacity of 40 men or one 8-ton locomotive. Hoisting ropes f cage and counterweight are 1-1/2-inch diameter locked coil with a breaking strain of 145.6 tons.

The Urquhart or main ore shaft has three compartments from the surface t no. 5 level. Below no. 5 level a sinking compartment was added (fig. 7).

The main ore hoist, built by Fraser & Chalmers (England), is a double, parallel-drum, single-reduction geared type with a Ward-Leonard control. Hoist operation is effected by manual or remote push-button control. The motor has a continuous rating of 1,525 horsepower at 258 to 480 r.p.m.

Two 9-ton, Kimberley-type, steel skips with Linatex bottom lining operate in balance at a maximum speed of 1,890 feet per minute. Ropes are the same type and size as those used at the supply shaft.

The Davidson and interior shafts (fig. 8) are duplicates in size and equipment. Hoists are single-drum, double-reduction, geared type with solenoid brake and limit switch controls. The hoist frame is structural steel, designed for assembly at place of erection. Drum castings are made i four segments to allow transportation through the 9- by 8-foot haulage level The motor is rated at 90 horsepower, 580 r.p.m., and gives a hoisting speed

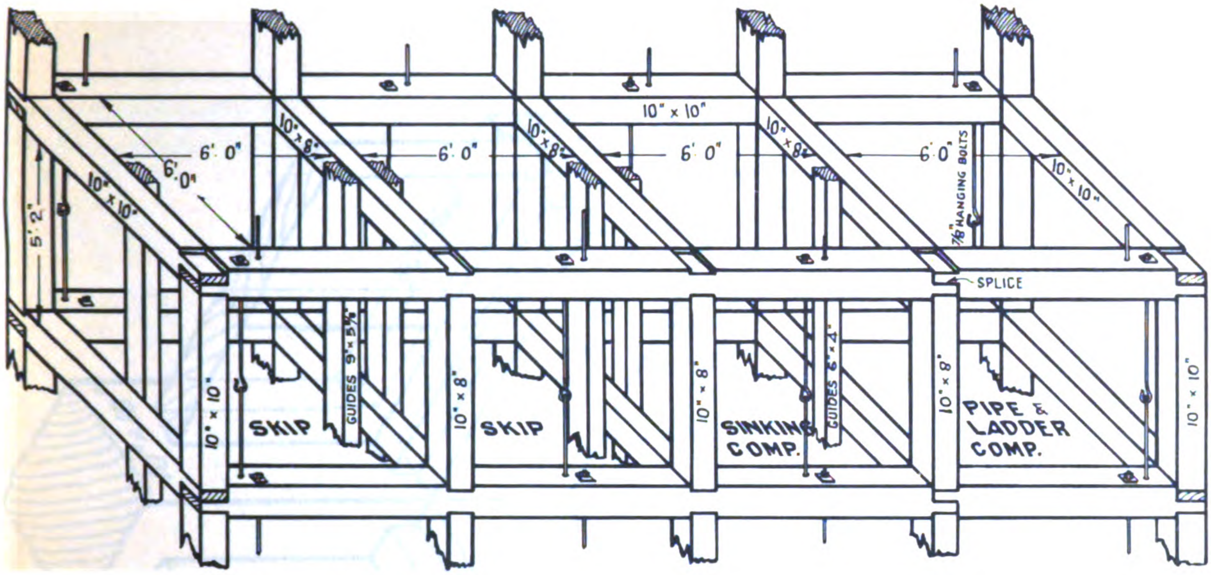


Figure 7.—Urquhart-shaft timber sets.

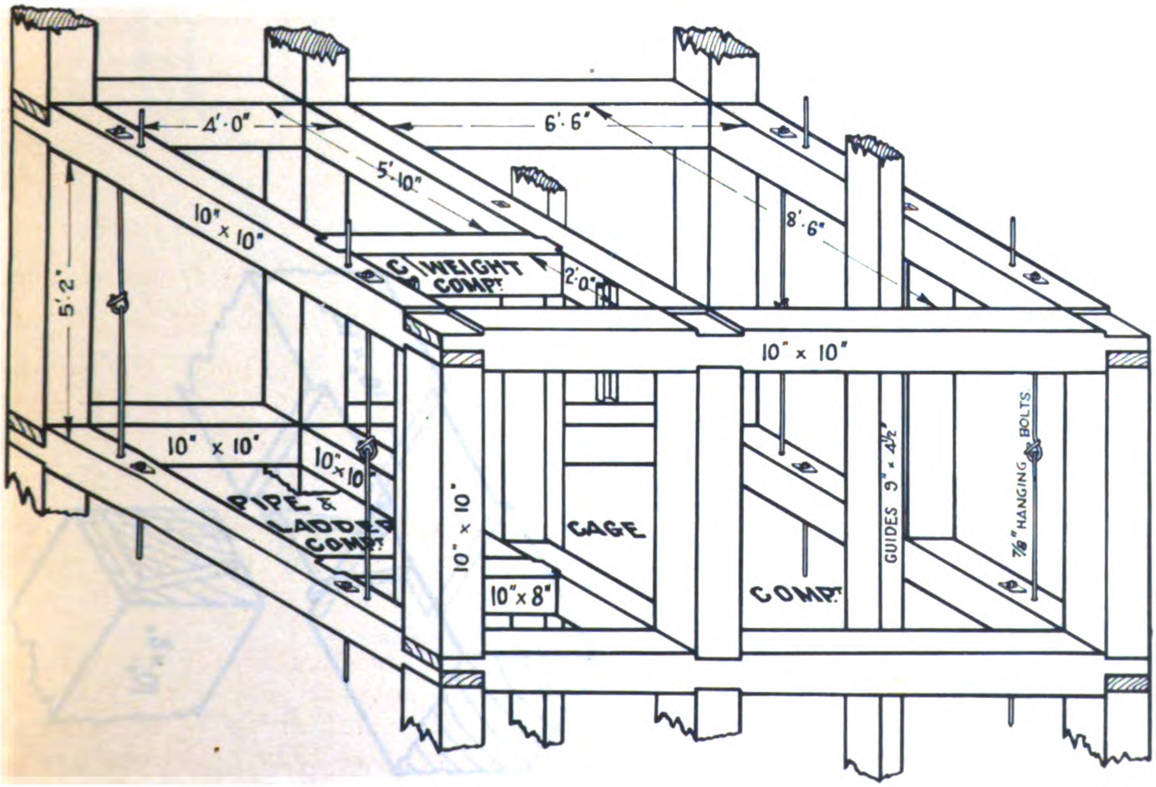


Figure 8.—Interior- and Davidson-shaft timber sets.

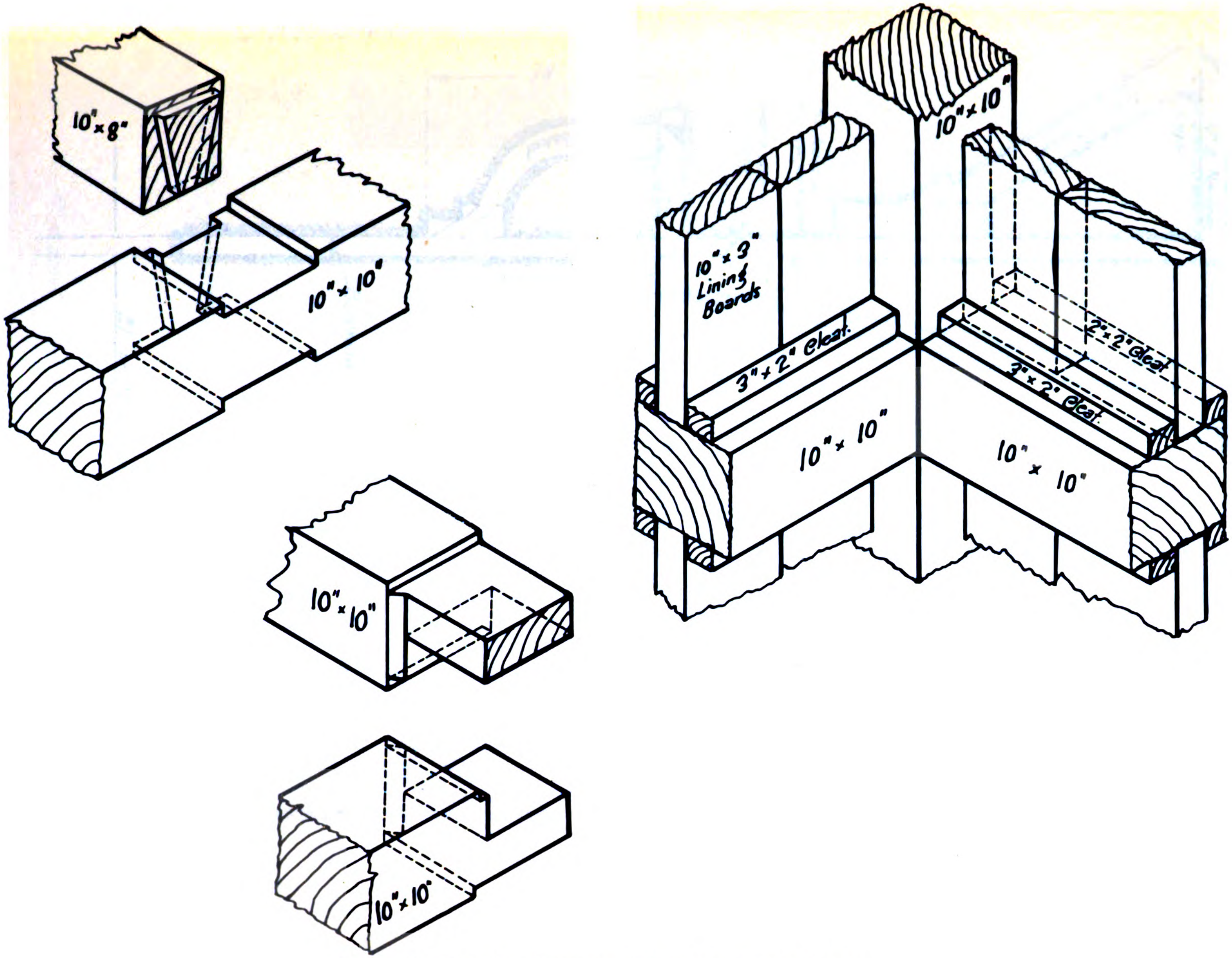


Figure 9.— Details of standard framing for shaft sets.

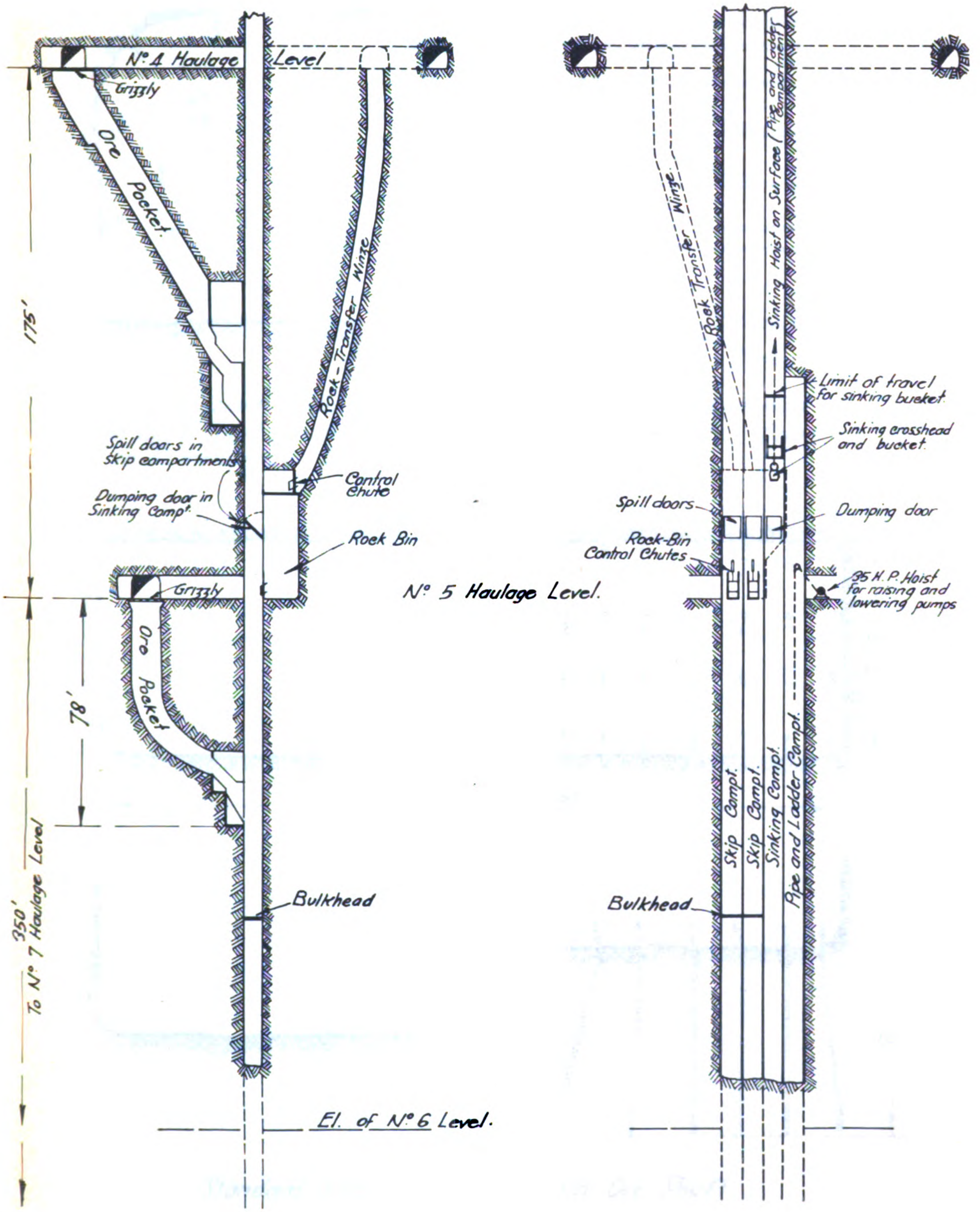
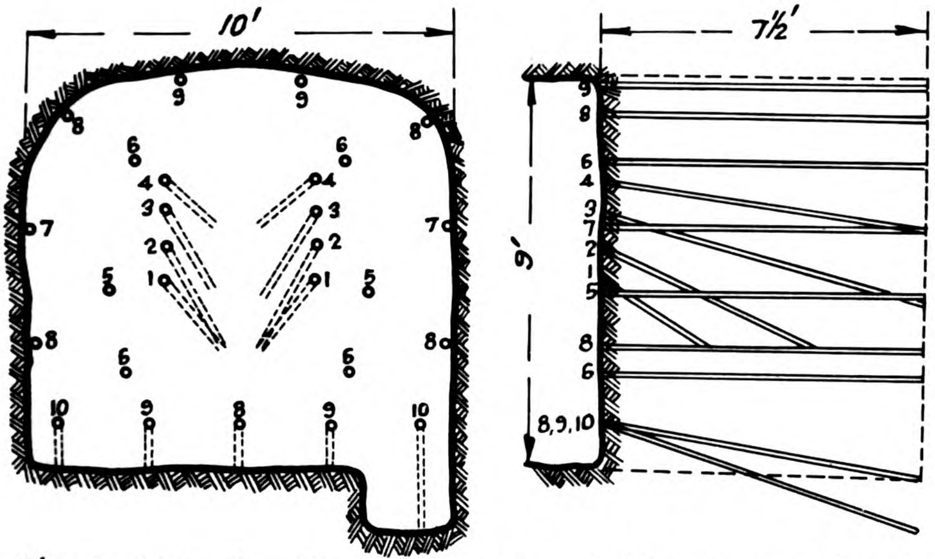
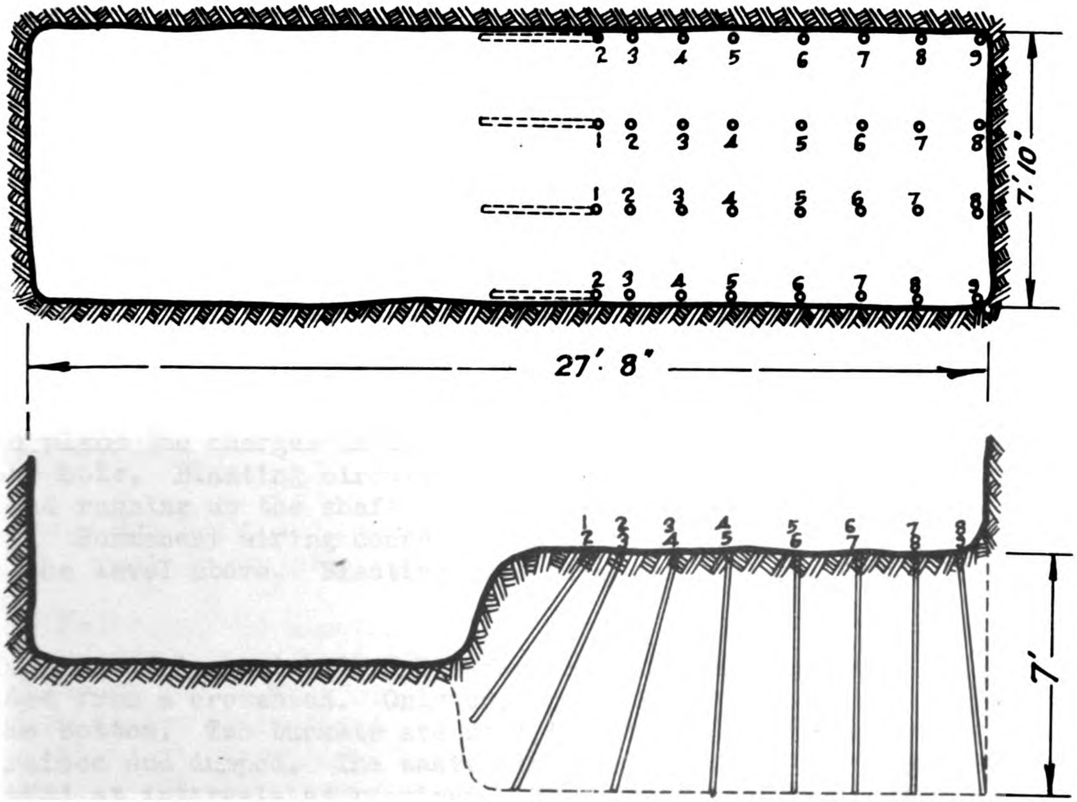


Figure 10.—Main ore-shaft sinking layout.



Standard Round – Main Haulage Cross-cut (Main Ore Shaft to Man and Supply Shaft.) 27 holes. Charge 110 lb. 60% gelignite. Delays 1 to 10 inclusive. Average break 7'.

A



B
Standard Shaft Round – Main Ore Shaft.

Note. Rotation of firing holes designated by numbers shown at collar of holes

Figure 11.—Firing rounds.

of 200 feet per minute. The cage is designed to take one 75-cubic-foot ore car and is counterbalanced by an 11,000-pound cast-iron weight. Cage and counterweight ropes are 1-1/4 inches and 1-1/8 inches in diameter, respectively.

The details of standard framing for all shaft sets are shown in figure 9.

Development details

Shaft and winze sinking

Shafts. - Arrangements for sinking the Urquhart shaft are shown in figure 10. The shaft penetrates barren shales varying in character from medium hard to soft, with some sections much broken and fissured. The soft fissured zones generally carry water to the extent of 200 to 600 g.p.m., with a temperature as high as 96° F. and rarely lower than 90° F.

The sinking crew comprises 1 shift boss, 6 miners, 1 platman (topman), and 1 pumpman. An extra pumpman is available on the day shift, when major repairs and changes are made and pipe lines extended. Varying conditions and water cause delays and prevent sinking operations from being maintained on a time schedule; each operation, drilling, blasting, mucking, and timbering, is done in sequence, as required, by the same men.

Drilling is done with heavy-type sinkers, using 1-1/4-inch round Leyner lugged-steel and 4-point cross bits. Three machines are used during the drilling operation, with the standard type of round illustrated by figure 11,B. Variations in the number and spacing of holes are required to suit changes in ground conditions.

Blasting is done electrically with delay igniters, nos. 1 to 10, and no. 8 lead azide detonators; the whole round or bench is blasted at one time. All holes are charged with 60-percent Gelignite. Water and ground conditions encountered throughout most of the distance sunk to date have made it necessary to place the charges in light galvanized-iron tubes, and the unit put into the hole. Blasting circuit consists of bell wire for connecting the holes and running up the shaft to permanent blasting lines at a short-circuiting box. Permanent wiring connects the short-circuiting box and the blasting box on the level above. Blasting is controlled and done by the shift boss in charge.

Mucking is done with 21-cubic-foot barrel-shaped sinking buckets suspended from a crosshead. Only one compartment is available for hoisting from the bottom. Two buckets are used, one being filled while the other is being raised and dumped. The waste bin opens into the skip compartment and is emptied at intervals as required. The rock is hoisted to the surface and discharged through an auxiliary rock chute.

Timber is generally kept close to the bottom. Sets are spaced on 6-foot centers, except when they pass through extremely soft broken-shale country.

In this type of ground it has been necessary at times to reduce the spacing to 4-feet. The bottom timbers are protected by a blasting set secured to the lowest timbers by chain slings. When timbering is done, the blasting set is lowered to the bottom, and a light set, suspended from the existing timbers, is lowered the required distance and used to support a working stage.

Pumping equipment of several types has been used, including air-operated Cameron sinkers and both large- and small-capacity, electrically driven, centrifugal sinkers. The small electric sinkers have been found to be the most suitable type, chiefly on account of their mobility. These pumps have a capacity of 200 g.p.m. when pumping against 100-foot head. They are connected in series when the required head to be pumped exceeds the capacity of a single pump. Pumping difficulties often are experienced because of the formation of very fine silt from the softer shale beds. This material is picked up by the pumps and packs in around the pump casing, valves, or other nonmoving parts until the pump is rendered useless. Washing back partly overcomes the difficulty, but after a period the hard-packed silt builds up to an extent that requires the pump to be dismantled and cleaned. Pumps are raised and lowered by a 35-horsepower, motor-driven, single-drum hoist.

The Davidson shaft below no. 5 level was enlarged to size and construction similar to that of the interior shaft, with the sinking arrangements as shown by figure 12. Conditions for disposal of ore and rock broken in the shaft favored the use of small 14-cubic-foot ore cars that fit into cages suitable for the original section of the shaft above no. 5 level. Extension runners were added to the original cages to permit lowering them below the timber. A mucking pan was constructed with a capacity slightly less than one full car. A 10-horsepower reversible hoist with control ropes extending to the shaft bottom is used for raising and lowering the mucking pan. The sinking hoist is on the surface, being the same hoist previously used for operating cages in the shaft. During timbering operations the mucking pan is removed and the hoist used to assist swinging timbers into position, thereby eliminating much of the heavy lifting and pulling ordinarily required to hang wall-plate and end-plate timbers.

Other details regarding drilling, blasting, and timbering practices are essentially the same as those described at the main ore shaft.

Winzes. - Most winzes are sunk for the purpose of completing connections to raises put up from the level below. Their depth rarely exceeds 75 feet. Those to be used as service ways for stopes are 7 by 5 feet in cross-section and where possible are sunk without timber. Inclination varies with dip of the lode, the average being 58° to 60°. The usual round employed in sinking winzes is shown by figure 13,A. Due to regulations contained in the Queensland Mining Act, a tugger or similar type air or electric hoist, when used for raising muck, must be controlled by a certificated operator or engine driver, who does nothing but operate the hoist. Men holding such certificates are comparatively scarce and cannot be obtained in sufficient numbers

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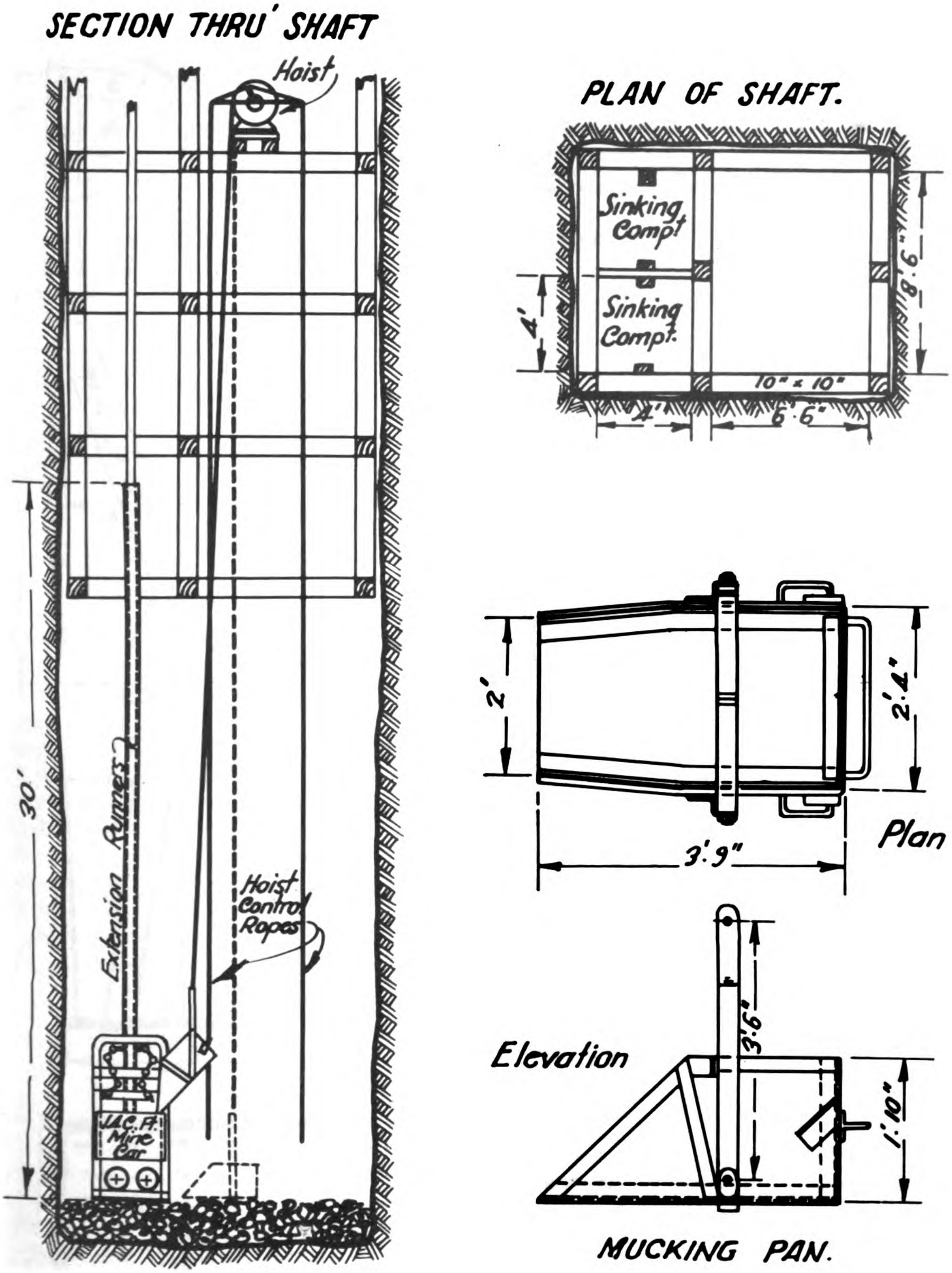
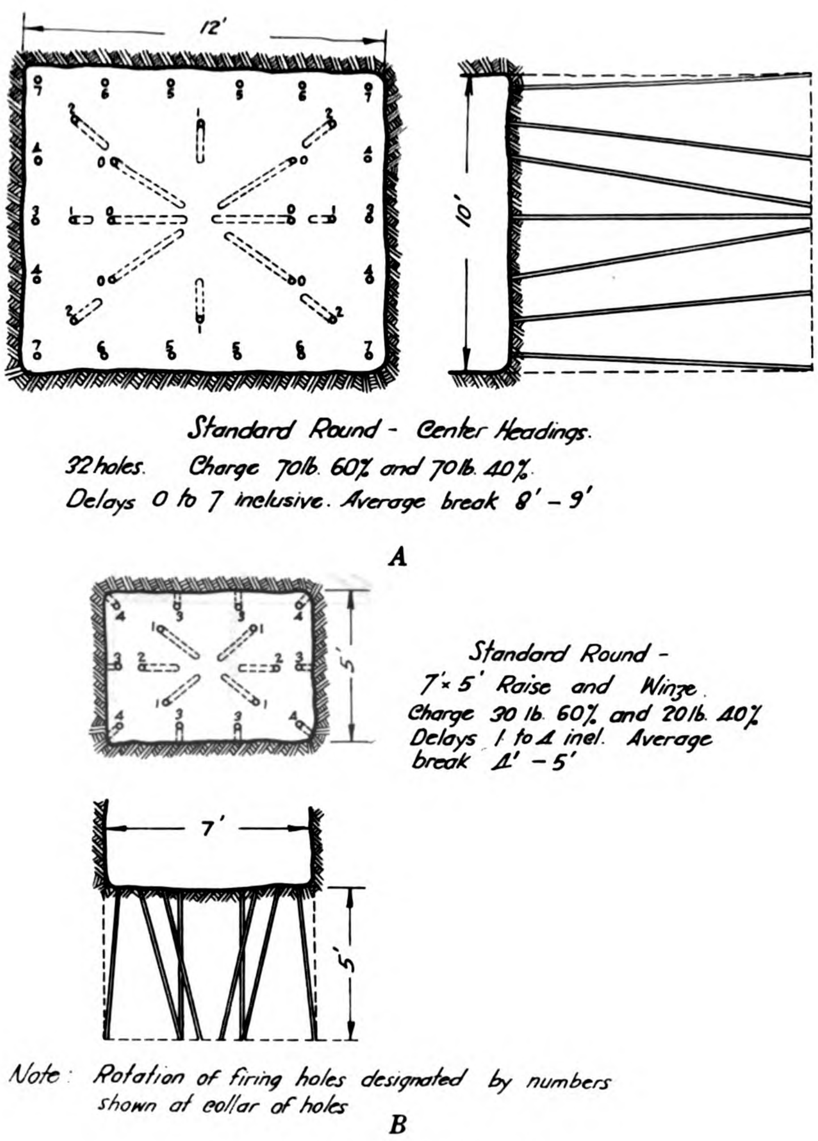


Figure 12.—Arrangement for mucking Davidson shaft sinking.



Standard Round - Center Headings.

32 holes. Charge 70 lb. 60% and 70 lb. 40%.
Delays 0 to 7 inclusive. Average break 8' - 9'

A

Standard Round -
7' × 5' Raise and Winze.
Charge 30 lb. 60% and 20 lb. 40%
Delays 1 to 4 incl. Average
break 4' - 5'

Note: Rotation of firing holes designated by numbers
shown at collar of holes

B

Figure 13.—Standard rounds.

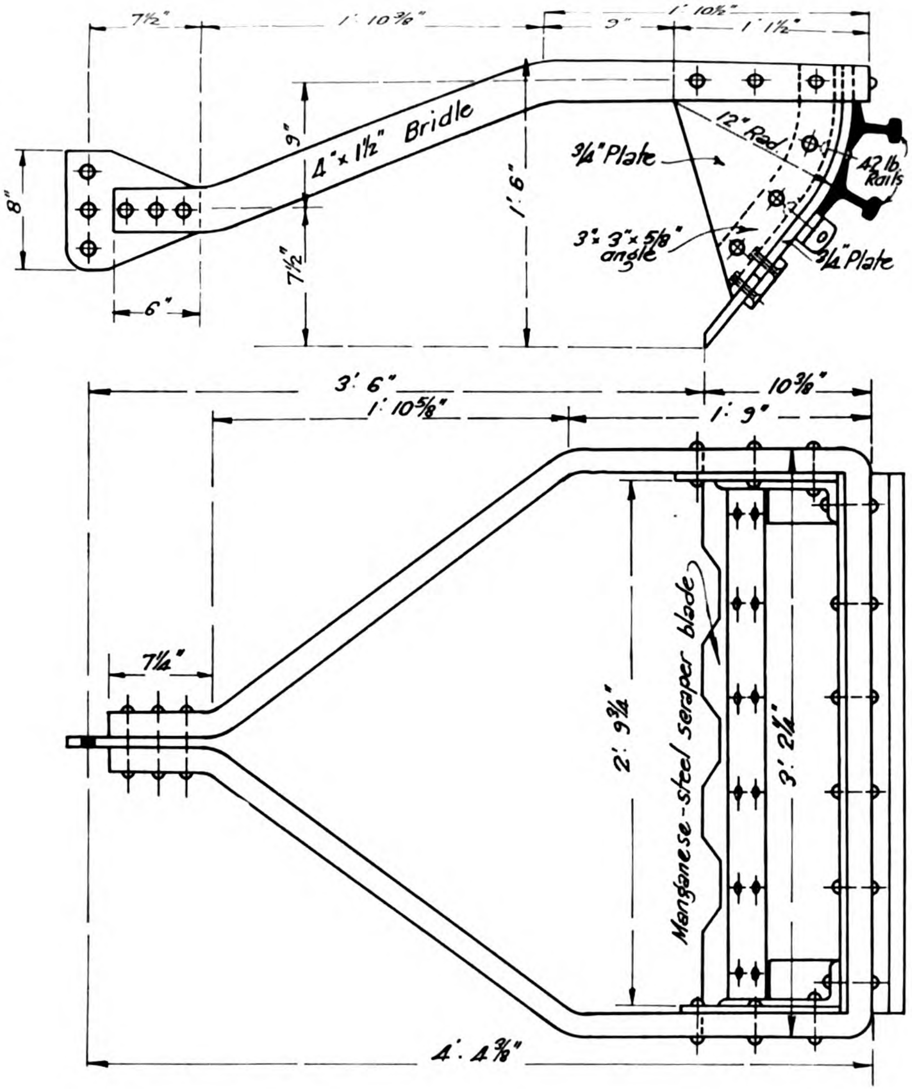


Figure 14.—34-inch hoe-type scraper.

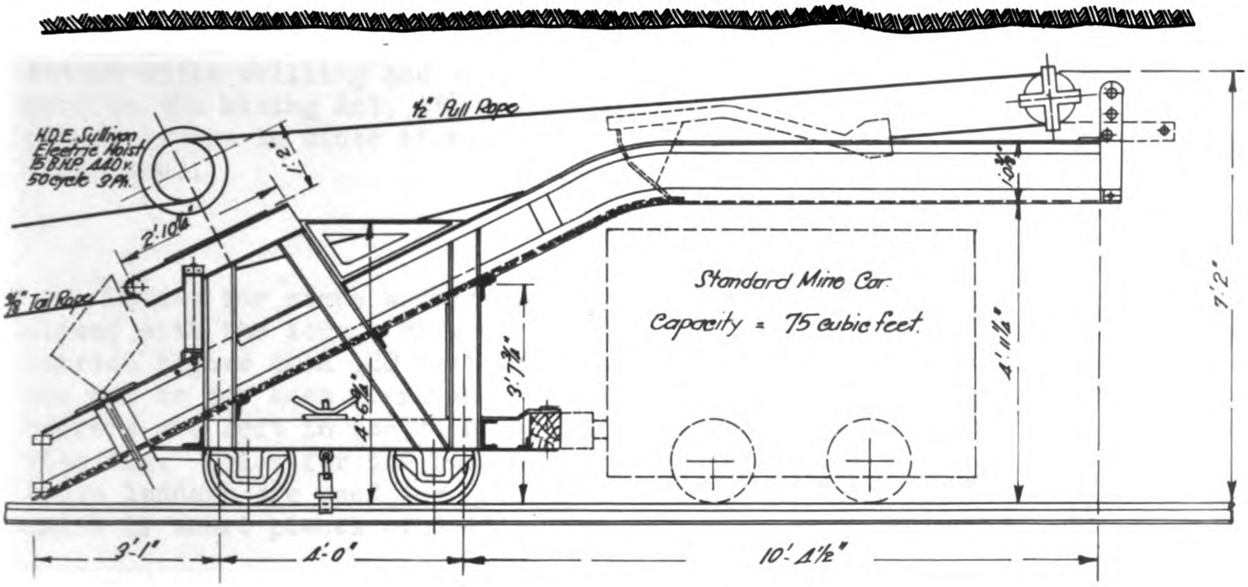


Figure 15.—Scraper slide mucking machine.

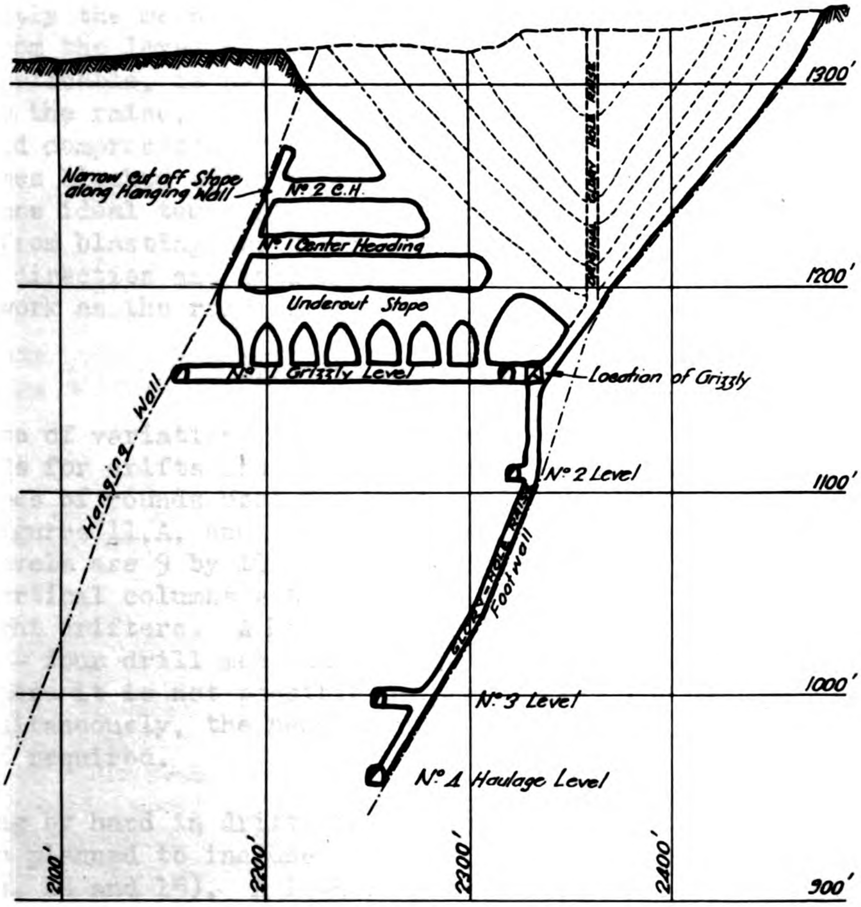


Figure 16.—Typical glory-hole section.

for winze work. In consequence, the old-fashioned windlass is employed for hoisting. A winze crew consists of three men, two of whom are on the bottom while drilling and on top while mucking. A recently proposed amendment to the Mining Act, if passed, will permit operation of a hoist by competent miners in winze sinking, with consequent improvement in winze-sinking operations.

Raises

Raises for stope service ways are 7 by 5 feet in cross-section and inclined with the lode. They are put up without permanent timber and are not carried higher than 120 feet above the working level. Stages for drilling are put in for each round and removed before blasting. Hangers for raise bearers are left in position to allow easy replacement of the bearers after blasting. Holes for the hangers are drilled as part of each raise round. Chain ladders are used for gaining access to the back and are secured in the raise by short pieces of drill steel.

In the past, ventilation of raises usually has been done by compressed air carried up in a drill hose or by 12-inch ventube carried up close to the face, through which air is supplied by a 2,000 c.f.m. auxiliary ventilating fan. Recently the method of ventilating raises through a diamond-drill hole put down from the level above was tried out with successful results, and wherever applicable, is now used in preference to other methods. After holing into the raise, a short length of pipe is concreted in the collar of the hole and compressed air blown down to the face of the raise. The hole often becomes blocked after blasting but can be opened without difficulty. By this means ideal temperature conditions are maintained in raises and smoke and fumes from blasting are blown out quickly. The hole serves, also, as a center for direction and inclination of the raise and thus saves much surveying work as the raise progresses.

Drifting and Crosscutting

Because of variations in drilling and breaking qualities of the ground, drill rounds for drifts and crosscuts are not standard throughout the mine. The two types of rounds used for most conditions with best results are shown by figures 11,A, and 13,B. All haulageways and drifts other than stope sublevels are 9 by 10 feet in untimbered sections. Drilling is done from two vertical columns set up at a clean face, by means of two heavy or medium-weight drifters. A heading crew in ordinary development work consists of six men - four drill men and two muckers - who work two headings alternately. Where it is not possible or practicable for two headings to be driven simultaneously, the heading crew consists of four men only, who drill and muck as required.

Mucking by hand in drifts or crosscuts is avoided wherever possible, and work is planned to include scrapers (fig. 14) operating on a loading slide (figs. 14 and 15). A 15-horsepower, electric, double-drum hoist with side-operating levers constitutes the power unit. Ropes are 1/2 inch in diameter and 9 by 9 by 1 construction. During the mucking operation the slides are serviced by 75-cubic-foot cars and battery locomotives. Track is carried as close to the face as practicable to minimize the scraping distance; it is also carried below grade to allow for ballast when finally brought to grade. Two types of air-operated mechanical shovels have been used under certain conditions with fair success, but they have not been suitable for loading into the 75-cubic-foot ore cars.

Headings are blasted electrically, the whole round being blasted at one time with electric delay igniters. Blasting is permitted only immediately before meal time or at the end of the shift unless a heading is in an isolated position so dust and smoke will not affect other workings.

Drilling equipment includes 3 1/2-inch-diameter, single-screw columns with Universal-type arms, either 3-1/2-inch or 4-inch drifters, 1-inch air hose, 1/2-inch water hose, and 1-1/4-inch hollow round steel with 4-point cross bits and Leyner lugged shanks.

Mining

Mining Oxidized Ores

Several mining methods were used in extracting oxidized ore, variations being required to meet local conditions. Inasmuch as these methods ceased with completion of carbonate-ore mining and had no features of peculiar interest, their description will be brief.

Glory holes. - The glory-hole system was used in mining most of the carbonate ore in the Black Star lode. The ore body in this section is lenticular and approximately 1,100 feet long on the long axis; the width varies from a few feet at the ends to 200 feet in the center. Both the enclosing shale beds and ore-bearing strata dip at an average of 55° to 60°, with barren overburden occurring only along the hanging-wall side, where it extended higher than the lode. This barren overlapping section was blasted and scraped away from the hanging wall prior to mining.

The area mined was divided into seven holes and a raise to serve as a draw-off pass put through for each hole from the haulage level through the grizzly level to the surface. Figure 16 illustrates a typical glory-hole section.

Mining was done by benching, working outward from the raise or pass, with sides carried as flat as practicable. Benches were carried around each hole, drillers working above the face left from the previous blast, thus reducing drilling and scaling work to a minimum. When a new bench was started around the bottom of the hole, it was first necessary to clean down all loose material on the sides so that work at the lowest elevation could be done safely. Drilling was done with medium-weight jackhammers using 1-inch hexagonal steel and 4-point crossbits with side holes. Holes varied in depth from 6 to 16 feet, according to conditions. All holes were sprung or chambered prior to being blasted. Several grades of explosive were tried, including black powder, but a slow-acting, 25-percent-strength ammonium-base explosive was found to give best fragmentation. Men were required to wear a safety harness with a rope attached to a steel peg driven in solid ground at the top of the hole. Ladders were placed down the sides at convenient places to make traveling and transportation of equipment easier and safer. Mining in the glory holes was done on day shift only.

Sublevel stopes. - After a glory hole was completed, a triangular-shaped section of ore remained below the hanging wall (fig. 16). To mine this section, alternate transverse stopes 30 feet wide and pillars 15 feet wide were laid out and sublevels driven from the sides of the glory holes to the lode hanging wall. Crosscuts were driven in the pillars on the grizzly level; draw-off raises and chutes were put up and each stope was completely undercut. A narrow cut-off stope was carried up the hanging wall, connecting each sublevel. The sublevels were ring-drilled (similar to the method now used in sulphide ore stopes) and blasted out as required. Rings were blasted with electric delay igniters. In some cases, where sublevels showed evidence of being difficult and hazardous to work in after blasting, the delays for each ring on the hanging wall and glory hole side corresponded. As many as 22 rings were blasted at one time with instantaneous detonators followed by delays nos. 1 to 10. The upper sublevel was comparatively short in all stopes and after one blast would have been both difficult and dangerous to reenter. Therefore, all rings in this sub were fired in one blast, starting with instantaneous detonators on the glory hole side.

When approximately 60 percent of the ore estimated to be mined by this method had been extracted, subsidence of the hanging-wall country occurred and crushed in the remaining stopes and pillars. Drawing of the crushed ore continued through stope chutes and glory-hole passes until dilution from barren hanging-wall material made it uneconomical to continue. Structural weakness of the country rock due to faulting across the lode was disclosed after subsidence. It was along these planes of weakness that the initial and greatest slumping occurred, the pillars between stopes proving too weak to withstand the weight placed against them after movement started.

Square-set stopes. - Oxidized ore in the Black Rock lode was mined in flat-back filled stopes and filled square-set stopes. The square-set method was applied to the upper half of the oxidized ore zone and extended through to or within a few feet of the surface. The ore in this section was weak and friable, with little or no bond between bedding planes, and required continuous support throughout any opening much larger than an ordinary drift. Square sets were of the post-butting type of 10- by 10-inch Oregon timber. Dimensions of sets were 6 by 6 by 6 feet. A surface belt-conveyor system carried mill-jig tailings along the top of the square-set stope section and discharged this fill material into raises. Filling was done as required, mining proceeding generally not more than two floors ahead of filling.

Flat-back stopes. - The flat-back filled stoping method was applied successfully to the lower half of the oxidized zone, where, due probably to incomplete oxidation, the beds were structurally stronger and the bond between beds was such as to allow stope openings without serious sloughing of ore or hanging wall. Where local weakness occurred along the hanging wall or back, support by stulls and headboards or by cribbed bulkheads was sufficient until fill was run in for additional support.

Methods on the Rio Grande lode. - The first stopes worked in the Rio Grande section were carried up as ordinary shrinkage stopes, but subsequent stopes along parallel panels were worked as inclined filled-rill stopes. The change in method was made because of excessive sloughing of weak hanging-wall beds as the shrinkage ore was withdrawn from stopes worked by that method.

Pillars remaining between the top of sulphide-ore stopes and the bottom of oxidized-ore stopes, which includes the mining level, are now being mined by vertical slicing, underhand, waste being drawn down from the surface to fill in closely after each slice is completed. The mucking and tramming level in this operation is made on top of the filled sulphide stope.

Mining Sulphide Ore

From beginning of production in June 1931 and until March 1935, mill feed consisted of approximately 75 percent carbonate ore and 25 percent sulphide ore. Of the total production for this period, 43.5 percent was cheaply mined glory-hole ore. On completion of the glory-holes, the mine was confronted with the problem of replacing this tonnage with sulphide ore from underground stopes. The major features considered in selecting a suitable stoping method were:

1. A low-cost mining method, owing to the comparatively low-grade nature of the ore.

2. A stoping scheme allowing development, stope preparation, breaking, and drawing of ore to proceed concurrently.

3. The tendency of Mount Isa ores to oxidize rapidly, with resultant adverse effect on mill metallurgy, favored a method of extraction in which broken ore could be kept at a minimum.

4. A plan for ore breaking, in which comparative safety in performing all phases of the system could be incorporated.

5. The elimination, as far as possible, of features depending on the mature judgment and ability of skilled miners, because of the comparative inexperience of the majority of underground workers available.

6. Though laminated in structure, the ore is hard and breaks in large blocks, thus eliminating the adaptability of a caving method of mining.

After thorough investigation and consideration of all features, the sublevel system with ring-drilling method of ore-breaking now in use was evolved by officers of the mining department as probably the safest and

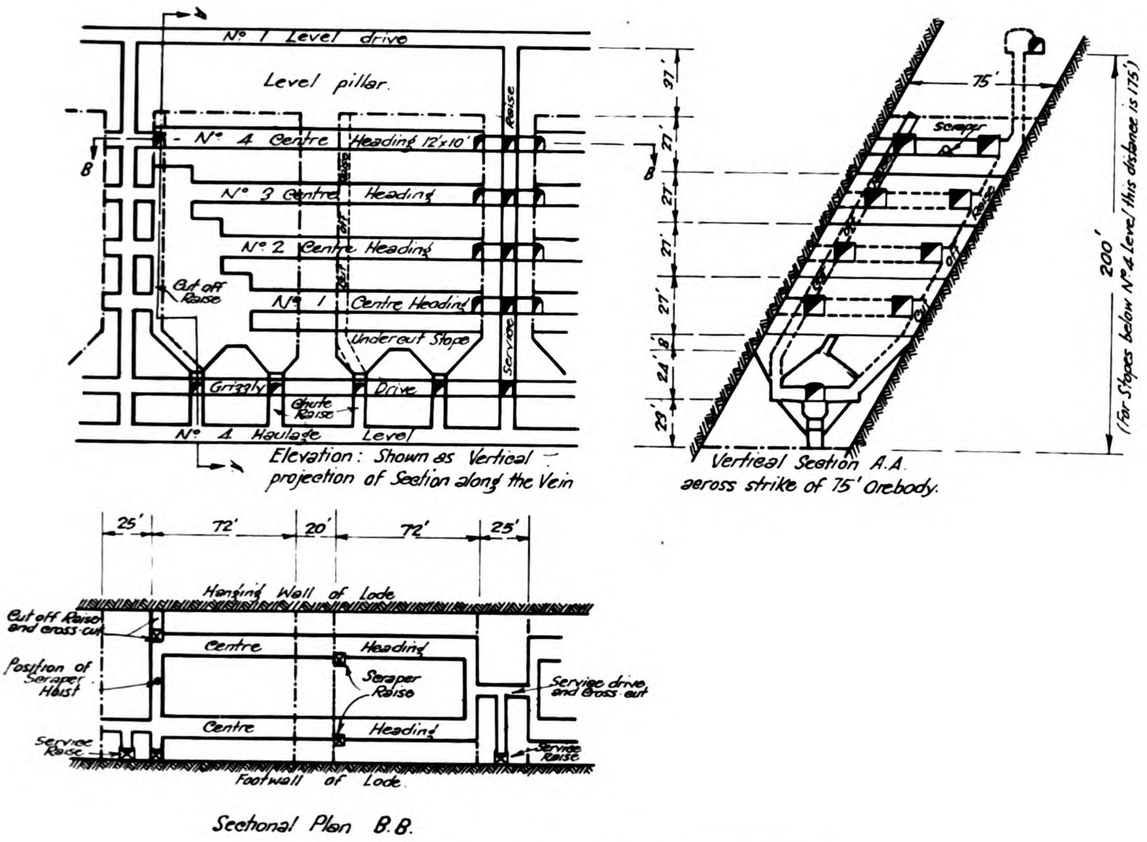


Figure 17.— Sublevel stope layout, longitudinal type.

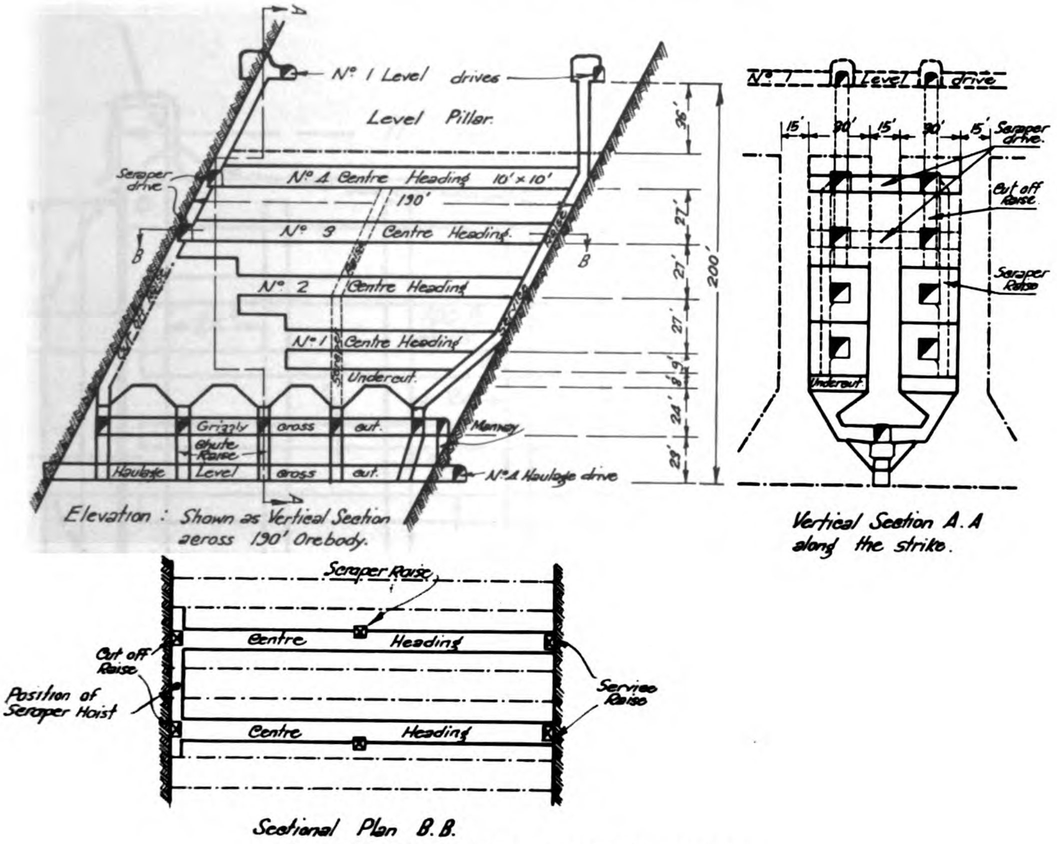


Figure 18.— Sublevel stope layout, transverse type.

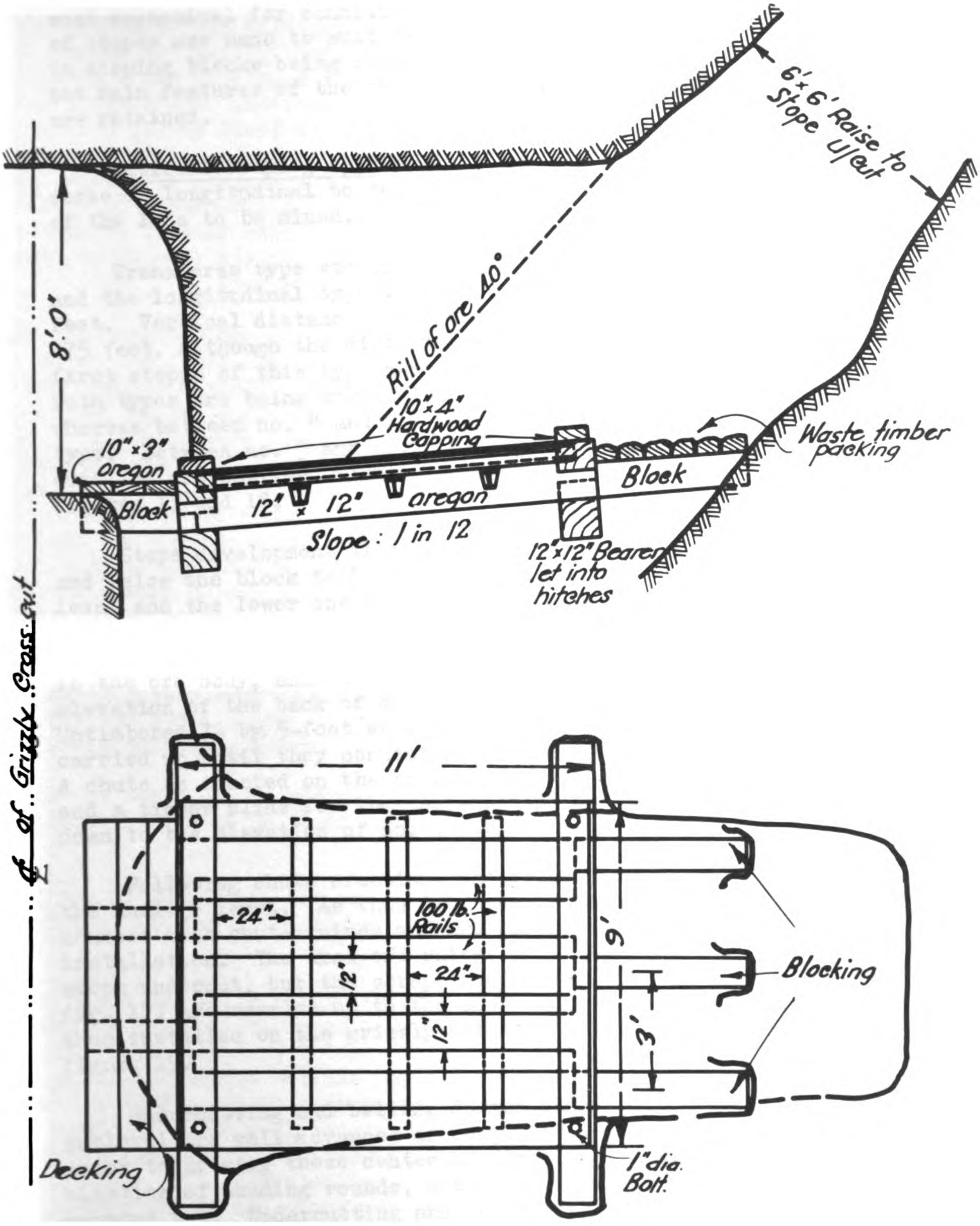


Figure 19.— Standard stope grizzly.

most economical for conditions prevailing at Mount Isa. Changes in size of stopes are made to suit ore limits and physical conditions encountered in stoping blocks being opened up from the lower haulage level at present, but main features of the system, method of development, and ore breaking are retained.

Sublevel stoping by ring drilling. - Stopes are laid out either transverse or longitudinal to the lode, the layout being governed by the width of the lode to be mined.

Transverse type stoping is applied where a lode width exceeds 75 feet, and the longitudinal type is used where a width varies between 30 and 75 feet. Vertical distance between mining levels has been established at 175 feet, although the distance between no. 1 and no. 4 levels, where the first stopes of this type were worked, is 200 feet. At present, stopes of both types are being worked in the section between no. 1 and no. 4 levels, whereas between no. 4 and no. 5 levels all stopes are of the longitudinal type. Between no. 5 and no. 6 levels both types will be again required to suit the ore outline. Ideal sections of both types are illustrated by figures 17 and 18.

Stope development is carried on from the levels immediately above and below the block to be mined; the upper level is the mining and service level and the lower one the draw-off or haulage level.

The haulage drift in the longitudinal type stope is located centrally in the ore body, and on its completion chute raises are put up to the elevation of the back of the grizzly drift and the chutes are erected. Untimbered 7- by 5-foot service raises are started from the level and carried up until they connect with winzes put down from the level above. A chute is erected on the haulage level and ladders, air and water pipes, and a timber slide for the tool-cradle are installed from the level above down to the elevation of no. 1 sublevel.

Following chute erection, the grizzly drift is advanced directly over the haulage level. As this drift advances, short crosscuts are put in to connect with chute raises and are enlarged to a size required for grizzly installation. The draw-off raises are extended up to the elevation of the stope undercut, but the scraper and cut-off raises (sectional plan BB, fig. 17) are carried up to the elevation of no. 1 sublevel. Grizzlies are then installed on the grizzly drift. A standard stope grizzly is shown by figure 19.

Undercutting and belling do not start until center headings on no. 1 sublevel are well advanced or completed. It was found that undercutting prior to driving these center headings left a section too weak to withstand blasting of heading rounds, and on two occasions large sections of the back dropped out. Undercutting and belling proceed simultaneously. A short drift is put through between two raises at the elevation of the undercut, from which horizontal and inclined stripping holes are blasted out until the wall limits are reached and the bell around each raise is formed. In stopes approaching 75 feet in width, a narrow supporting rib is left along the center line of the undercut to act as a temporary pillar for the no. 1 sublevel. Little or no shoveling of broken ore is required, and a good production per man shift is obtained from this operation. All other development and stoping operations are done from the level above.

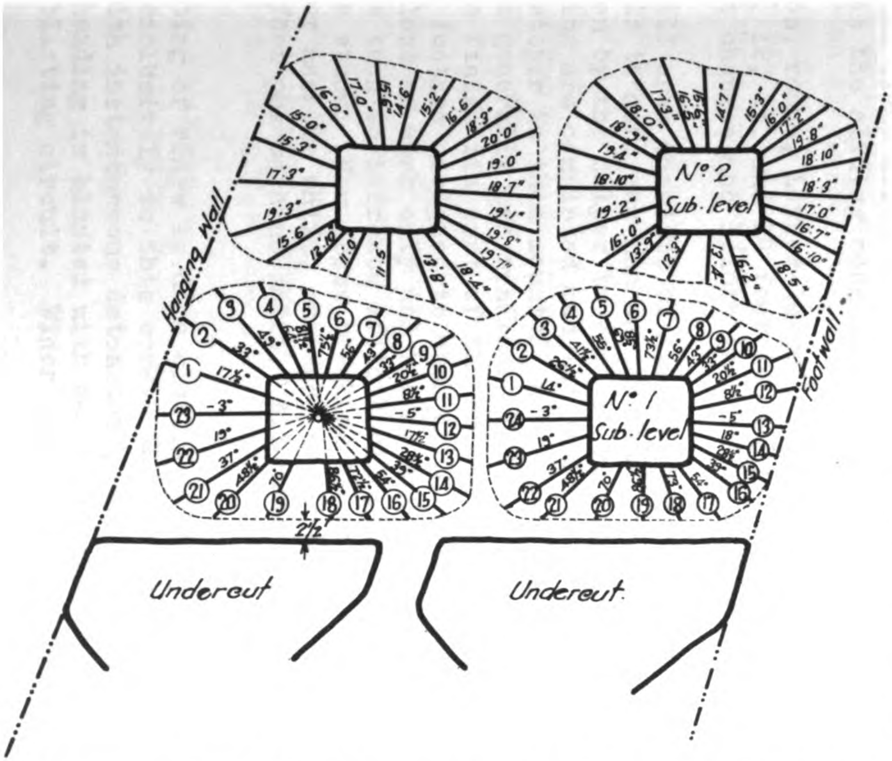
At the elevation of each sublevel a short crosscut into the stope pillar is started from the service raise. A drift is extended from this crosscut to the edges of the pillar, and along each side thereof crosscuts are put in to the lode limits. Center headings are driven from these crosscuts, which later serve for mining out the cut-off stopes.

A heading crew has six men: two machine runners, two machine helpers, one scraper driver, and one helper. One crew drives the two headings on each sublevel. The standard drill round for a 12- by 10-foot heading is shown by figure 13,B. Drilling is done from vertical bars, using 3-1/2-inch-diameter Leyner drifters. All rounds are blasted electrically. Drilling and mucking alternate between the two headings. Mucking is done with a 15-horsepower electric scraper-hoist and 34-inch hoe-type scraper. The hoist is in the cut-off crosscut, mounted on a turntable to permit reversing as required to scrape in each heading. Extra sheaves are used to carry scraper ropes around the corner of the crosscut into the heading. The cut-off raises at the end of the stope serve as transfer raises for ore scraped out of each heading, until the heading has advanced to the position where connection is made with the scraper raise. The scraper raises then become the mucking transfer, the object being to keep scraping distance at a minimum. On completion of each heading the scraper crew make a final cleanup of broken ore, while the drill crew extend the cut-off and scraper raises to the elevation of the next sublevel and continue heading work.

The cut-off stopes at the end and in the center of the stope are mined out by a separate crew, their operations being timed so as not to conflict with heading work.

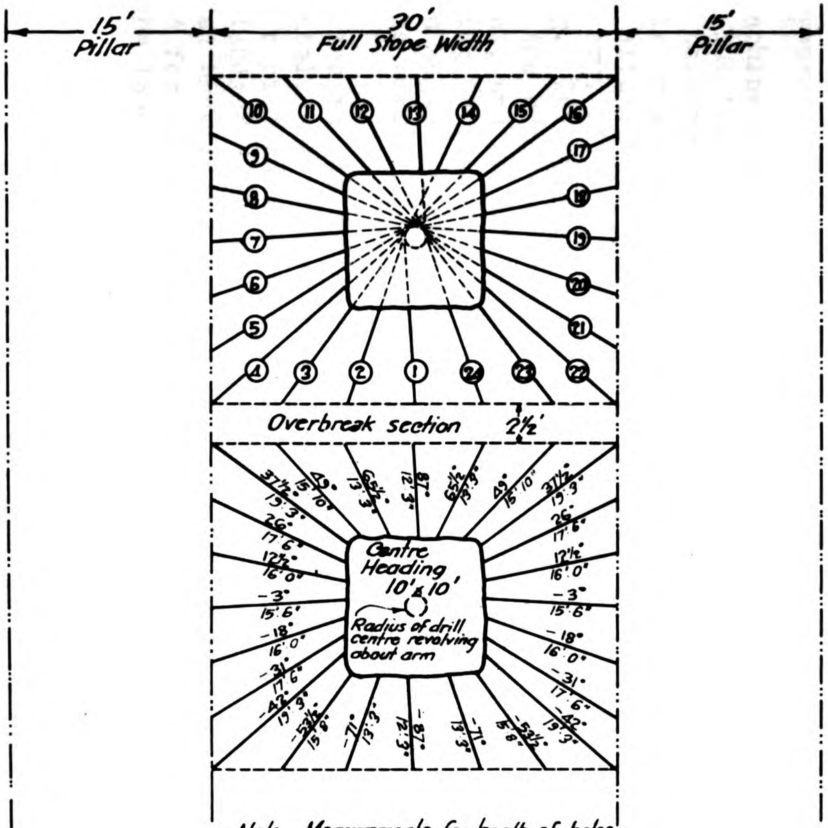
The ring-drilling system of breaking ore is the adaptation to mining of a similar system successfully employed by railway contractors to enlarge pioneer tunnels to full size.

Ring-drilling (figs. 20 and 21) is started after no. 1 and no. 2 sublevel headings have been completed and heading crews are working on no. 3 sublevel. The service raise on the opposite end of the stope from which headings are driven is used for traveling and service by ring-drilling crews and blasting crews, each raise serving two adjacent stopes. When a heading is ready for drilling the surveyors paint a center line along the back of the stope, a horizontal elevation line along each side, and short vertical lines across the elevation lines at proper ring interval. The distance between rings is 4 or 5 feet, depending on the character of the ground. Drilling of rings starts at the cut-off stope and works back to the service raise pillar.



Average Footage drilled for Nº 1 Sub-level rings = 250' per ring.
" " " " Nº 2 " " " = 244' " "
" Tonnage broken per ring for Nº 1 Sub-level = 372 tons
" " " " " " Nº 2 " " = 380 "

Figure 20.— Black Star longitudinal sublevel stopes, ring-drilling diagram.


15' Pillar
30' Full Stope Width
15' Pillar
Overbreak section
2½'
Centre Heading, 10' x 10'
Radius of drill centre revolving about arm

Note. Measurements for length of holes taken from centre point of heading

Total footage drilled per Ring - 247'
Tons broken per 4' Ring - 291
" " " 5' " - 358

Figure 21.— Black Star transverse sublevel stopes, 24-hole drilling diagram.

Machine mountings are 3-1/2-inch-diameter bars with universal-type arms. The bar is set up vertically midway between two rings to be drilled in the heading, so that the arm, when clamped to the column, will be directly under and parallel to the painted center line. In elevation the arm is situated so that the center line of the machine, after it is clamped on the arm, will be level with the elevation marks on each side of the heading. This places the machine so that its axis rotates in a circle around the arm, corresponding to the plan from which the hole pattern is laid out, and at 90° to the heading center line. Each hole is drilled according to the plan, the driller setting the machine accurately at the desired angle by using a small box-type inclinometer designed specially for this work.

Holes in the bottom half of the ring, including those slightly above horizontal, are drilled with wet-type Leyner drifters that have 4-inch-diameter pistons and special sliding cone shells. The shell of the drill is a standard 30-inch sliding-cone type with the slide for the cone clamp extended back to the crank end. This allows an extra long run of steel to be used for drilling any part of the hole and is particularly advantageous when the long finishing steels are used.

The remaining holes are drilled with wet-type, automatic, rotated stopers that have 3-inch-diameter pistons, reverse feed for bar mounting, and chucks for Leyner lugged steel. The construction of the reverse feed cylinder and clamp allow it to be used to get extra long runs on a drill the same as the sliding cone on drifters.

Hollow, round, Leyner-lugged, 1-1/4-inch steel in lengths of 3, 5, 7, 9, 10, 11, 12, 13, and 14 feet is used. Decrease in bit gage is 1/8th inch for 2-foot changes and 1/16th inch for 1 foot changes.

A drill crew consists of two drill-runners and one helper. Usual practice is to have one man on the drifter and one on the stoper, assistance being given by the helper to each as required. Holes drilled by the drifter in each ring are completed and the machine is moved along to the next set-up. The stoper is then mounted on the same bar and the ring is completed. Three bars generally are available for each crew to obviate waiting for one machine to finish its part of the holes. Actually, the time required to drill the footage assigned to each machine is nearly equal. Crews are kept on this class of work only and become specialized in ring drilling. The result has been to increase efficiency, drill holes accurately, and to have few steels stuck. New crews breaking in on ring-drilling average only 40 to 50 feet per machine shift, whereas, after they are experienced, they average 60 to 90 feet per machine shift. All drilling is done on contract-footage basis.

Blasting of rings is done electrically, crews of blasters being assigned exclusively to this work. The ring in the hanging-wall heading is blasted with instantaneous detonators, and the corresponding ring in the footwall heading is blasted with delay detonators, both being hooked into the same blasting circuit. Wider distribution of explosives in the hole is obtained by using 1-1/4- by 8-inch wooden spacer plugs. Blasting on each heading is kept in step so that no heading is blasted more than 4 rings in advance of the heading above.

Explosives used are 40 and 60 percent gelignite. In several stopes now practically mined out, explosive consumption is 0.40 pound per ton broken for primary blasting and 0.027 pound per ton drawn for secondary blasting. Variation in consumption is due chiefly to secondary blasting, depending on the hardness of the ore, and in some sections has been as much as 0.11 pound per ton drawn.

This method has given extremely good results in breaking to wall limits, though it is necessary to make close inspection of rings blasted on no. 1 sublevel to make certain breaking extends out to the ore limits. If the break from these headings is inside the limits, succeeding heading rings will not break past the walls so established and much ore will be lost. With ore distributed irregularly along foot and hanging walls the method would not permit complete extraction, but with even distribution of mineralization, as encountered in the Black Star lode, comparatively complete extraction with little or no dilution is effected.

Consideration has been given to the economy of driving the 12- by 10-foot headings, as compared with smaller headings; the larger headings have been retained for the following main reasons: (1) The large heading is required for subsequent ring-drilling operations. (2) A large heading gives greater tonnage per man shift than a small heading. In order to drill the longest holes required for a ring it is necessary to have at least 12 feet horizontal measurement in the heading. A lesser distance causes much trouble and lost time in entering long steel in a partly drilled hole. Smaller 10- by 10-foot headings were used in the transverse type stopes where the total width to be drilled was 30 feet, horizontally, but in the longitudinal type stopes the drilling from a single heading must reach 35 feet, horizontally. The drilling and mucking cycle in 12- by 10-foot headings works out so that there is no delay between the two crews that perform each operation. The rate of advance has consistently averaged 0.5 foot per man-shift, giving a production for stope development heading work of approximately 5 tons per man-shift. With headings smaller in section than 12- by 10-feet the footage advance per man shift is approximately the same and the resulting tonnage per man-shift is reduced correspondingly, with consequent increase in the cost per ton of ore produced from such work.

Training crews for one class of work and keeping them continually on that work has increased the general efficiency and safety of all operations.

Direct breaking cost per ton of ore into the haulage-level chutes, for several transverse-type stopes completely mined, are given in the following table:

	Labor	Explosives	Total
Stope development	$0.309	$0.096	$0.405
Construction of chutes and grizzlies [1]	.034	--	.034
Undercutting and cut-off stopes	.076	.027	.103
Ring-drilling	.115	--	.115
Ring-blasting	.016	.073	.089
Grizzly-breaking	.045	.007	.052
Totals	.595	.203	.798

[1] Includes timber.

Sublevel stoping by benching. - Sublevel stoping, as applied to the comparatively narrow lodes of the Rio Grande section when ore widths are 6 to 14 feet, differs chiefly from the method already described in that ore is broken from sublevels by slashing and benching instead of ring-drilling and blasting.

Opening stopes, driving sublevels, and other development is done in similar manner to methods used for the larger stopes. Sublevel drifts are 7 by 5 feet in section, with a vertical distance of 30 feet between floors.

The first level, termed the "undercut drive", is opened out to wall limits and drawoff raises coned out to form the stope undercut. Mining commences on the first sublevel above the undercut, at a scraper raise approximately midway between stope serviceway pillars and retreats towards the pillars. Where ore width requires, stripping to wall limits is done first by means of a bar-mounted drifter machine. Horizontal holes are drilled to strip a bench 6 to 8 feet wide, the height of the sublevel drift, and extending to wall limits. These holes break toward the center scraper raise; the resulting bench is cleaned off and drilled with down holes spaced at 3-foot intervals. Each bench is carried down until it breaks into the stope below. Generally, three such benches are carried down between sublevels, the last one is roughly 12 feet deep. Holes are not drilled above the sublevel, which leaves the back available for careful scaling before starting down with the benches.

Bench-drilling is done with heavy, sinker-type jackhammers; stripping or slashing is done with 3-1/2-inch-diameter mounted drifters. The jackhammers use Leyner lugged steel, so that the one type of steel is suitable for both machines.

Safety belts are worn by all men working on the benches, the rope being fastened securely in the sublevel drive.

Underground Transportation

All broken ore runs by gravity to chutes on the haulage levels, where it is withdrawn and transported an average distance of 3,000 feet to the main ore-shaft. While carbonate ore was being mined, subsidiary tramming from stopes to glory-hole or main transfer passes was done by hand with 14-cubic foot cars. Since completion of this work, hand-tramming of ore has been eliminated.

Haulage-Level Chute-Control

Two standard types of chutes are used for draw-off and transfer raises (figs. 22 and 23). The single type chutes are used where stoping width is comparatively narrow and broken ore gravitates to a single raise. The double type chutes are used where stoping width requires two raises for efficient drawing, or where raises serve adjacent stopes, as in the transverse stopes. All timber for chutes is framed on the surface and is ready for erection when sent underground. The arc-type gate (fig. 23) is operated easily by hand and effectively controls the flow of ore. With few exceptions, the gates outlast the life of a stope and are therefore salvaged and used over again. Where chute timbers are subject to wear they are protected by steel plate or by 2-inch local hardwood. This wood is highly resistant to wear and makes an ideal lining material.

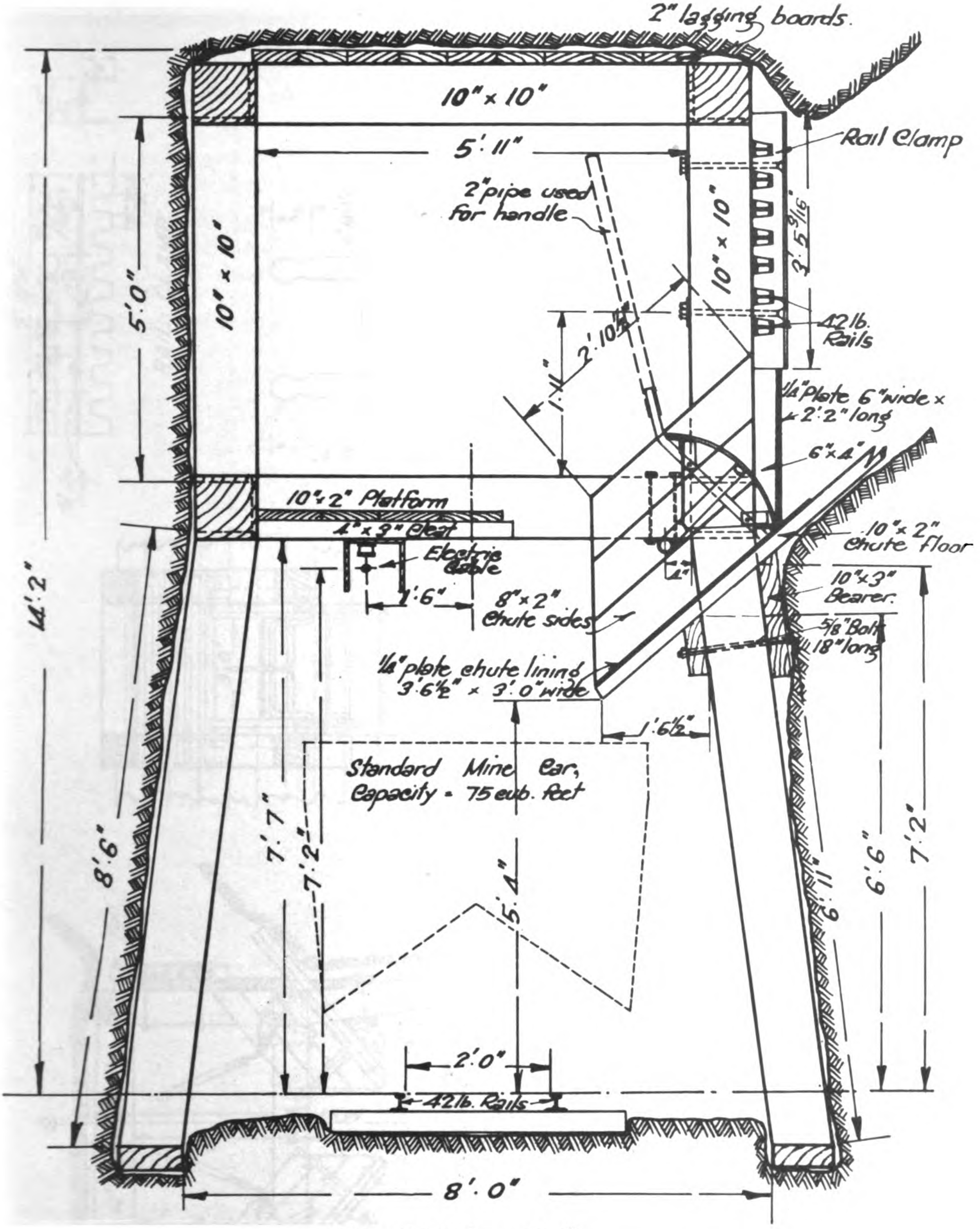
Tramming Equipment

Cars. - Ore cars are gable-bottom, side-discharge type (fig. 24), having a capacity of 75 cubic feet, or approximately 3.9 tons of broken ore. Wheels are cast-steel with Timken tapered roller-bearings and, except for moderate flange and tread wear, give excellent service. Drawbars are spring-cushioned with semiautomatic jaw-type couplings.

Locomotives. - Four 8-ton trolley locomotives, each driven by two 40-horsepower 460-volt, direct-current motors and manufactured by the English Electric Co., Ltd., are in service. They are equipped with both magnetic-control and hand-operated brakes. A normal train consists of 20 loaded cars and operates at a speed of 6 to 10 miles per hour. Trailer storage battery units of a size and capacity sufficient for full motor duty, were made up for the trolley locomotives; alterations made in controller connections permit quick changing of power from the trolley line to the battery unit. Locomotives engaged in loading, gathering, or shunting in stope crosscuts or drifts where overhead power is not installed obtain their power from a trailer battery unit. The complete battery consists of 117 cells, lead-plate type, having a capacity of 222 ampere-hours.

Four 4-1/2-ton and two 2-1/2-ton Mancha storage-battery locomotives are used for servicing development headings for minor ore or rock haulage and for general transportation of explosives, steel, timber, and other supplies. A spare battery is available for each of these locomotives.

Power. - Power is obtained from a 150-kw. motor-generator set situated on the surface. Transmission underground to distribution switches is through 0.25 square-inch, twin-core, lead-covered, armored cables. The total footage of trolley wire installed on no. 4 level, approximately 6,900



Width of Chute = 3' 0"

Figure 22.— Standard single-type chute.

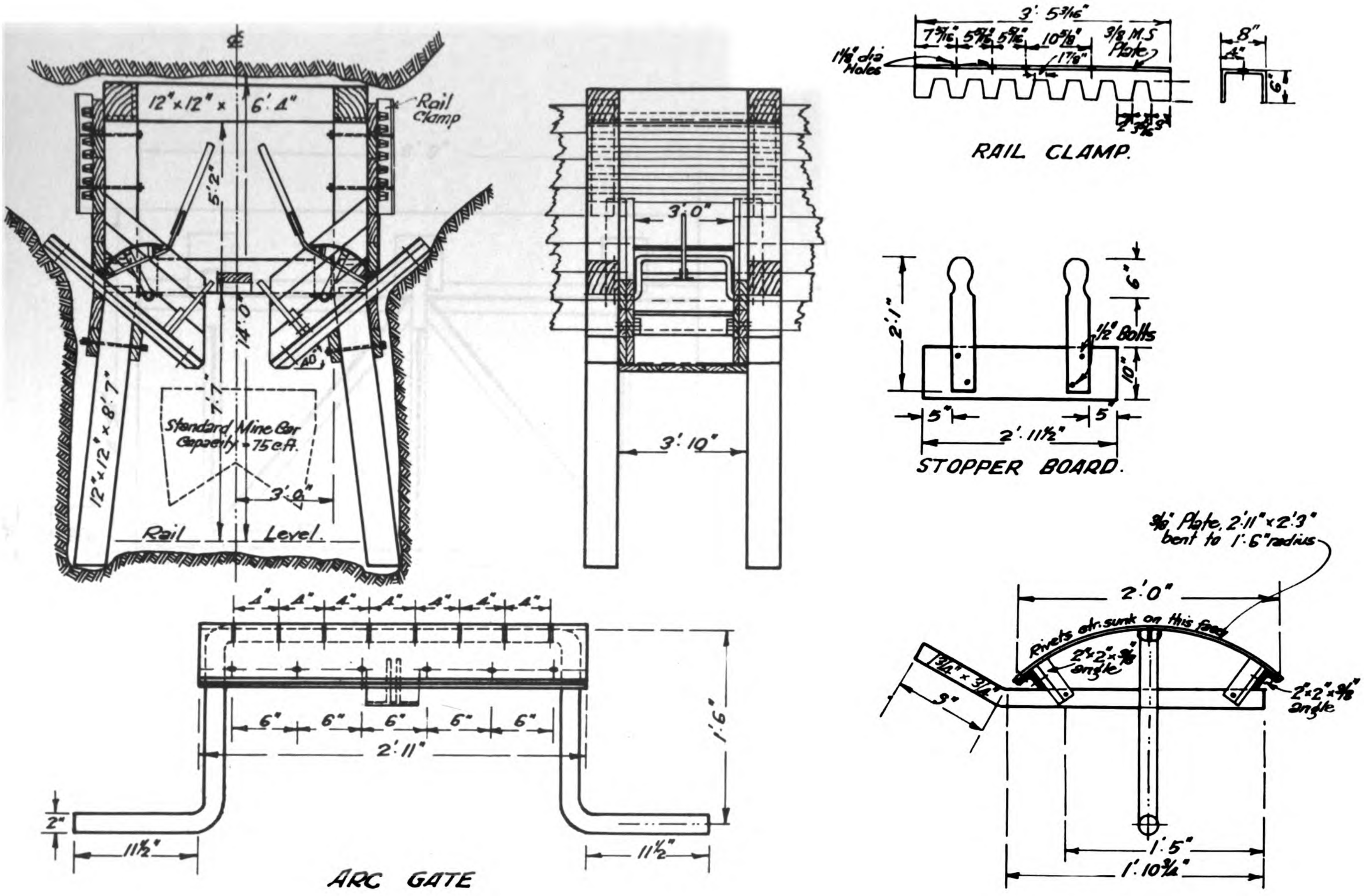


Figure 23.— Standard double-type chute.

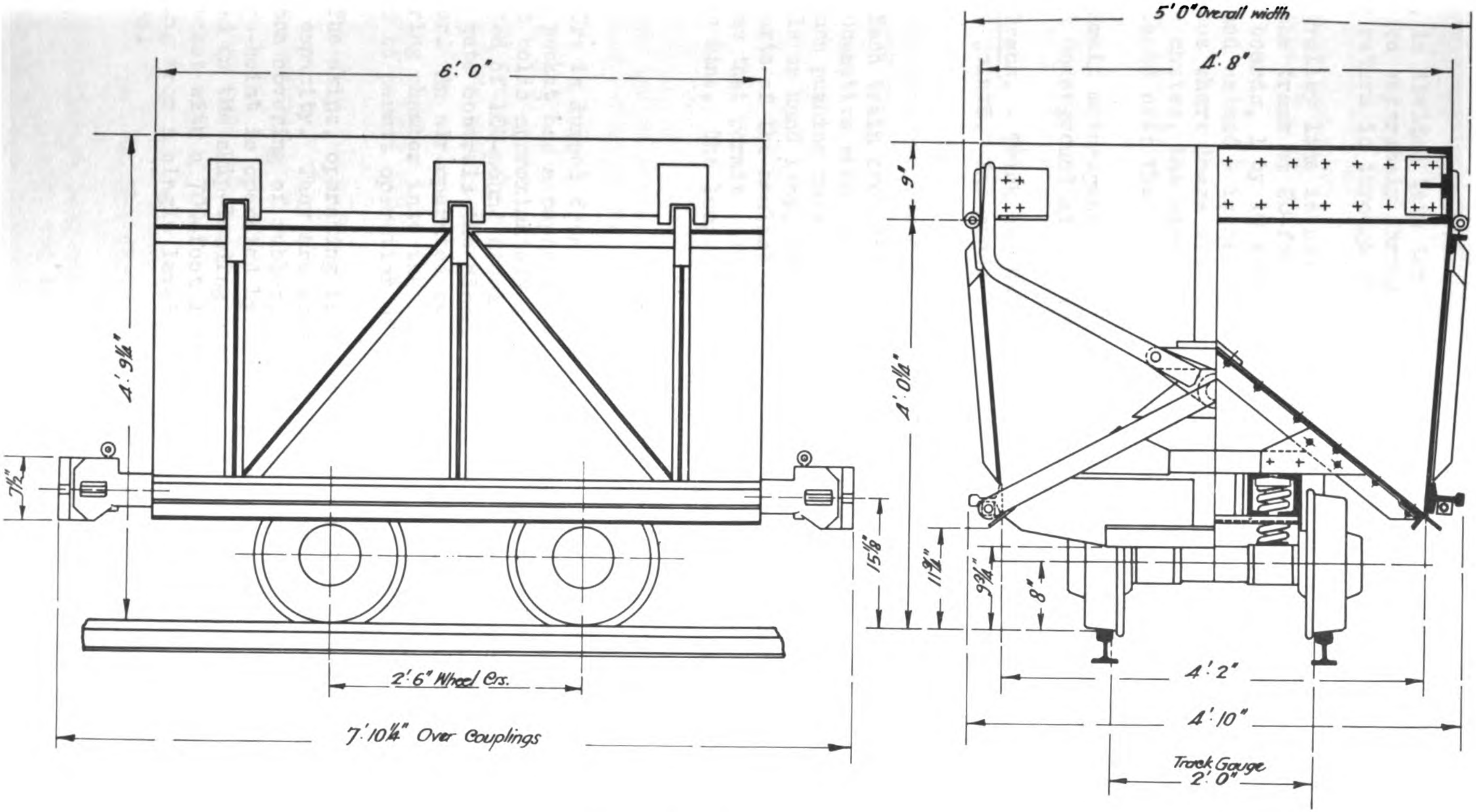


Figure 24.— 75-cu. ft. mine car (weight, 4,980 lb.).

feet, is divided into three sections, and power for each section is supplied separately through 0.15 square-inch, single-core, feeder cables. Power return is through the rails.

Trolley line is no. 4 - 0 grooved copper wire, supported centrally over the track at 20-foot intervals and 7 feet above the rail level. Guard boards, 1 by 10 inches, are placed 4 inches on each side of the wire and extend 4 inches below the wire. No trolley wire is installed in sections where there are duplex draw-off chutes. When it passes under single chutes, the wire is offset 18 inches, and additional guard boards are placed over the wire for a short distance on each side of the chute.

Small motor-generator sets for charging storage-battery units are placed underground at the man and supply and the interior shafts.

Track. - Track is 24-inch gage and consists of 42-pound rail with bonded joints. The grade is 0.5 percent in favor of the loaded trains.

Tramming Operations

Each train crew consists of a motorman and brakeman. Both men ride the locomotive when pulling cars. The brakeman rides in the next to last car when pushing cars. Signals are transmitted by the brakeman with a whistle or hand lamp, or both. A red trip lamp is carried on the last car. All parts of the haulage level are equipped with hand-operated block-signal systems that permit safe operation of two or more trains in the same section of the mine. The locomotive is always kept on the down-grade end of the train.

Ore hoisting

Ore is dumped from cars into pockets at the main ore-shaft. The no. 4 level pocket has a capacity of approximately 500 tons; the no. 5-level pocket holds approximately 300 tons. Grizzlies over the pockets are constructed of 100-pound steel rails with 14-inch spacing. An air-operated chain gate controls the discharge from the ore-pocket into a measuring-chamber. An air-operated over-cut door controls the outlet from the measuring chamber into the skips. Levers for operating each gate are interlocked to permit operation of either gate only when the other one is closed.

The skips, operating in balance, are self-dumping Kimberley type of 9-ton capacity. They are constructed of steel with removable wearing plates. A bottom covering of rubber, known as Linatex, prevents fines building up. The ore-hoist is operated by the skipman by means of pushbutton controls located on the skip-loading station. Hoisting capacity from the no. 4 level ore pocket with a 700-foot lift is 250 to 300 skips per 8 hour shift. When hoisting from a single level the capacity exceeds the maximum rate of haulage.

Extraction

Extraction for sulphide stopes cannot be given, as no section is as yet completely mined out. Figures for the sublevel system described show approximately 60 percent of the total tonnage as extractable during the stoping period. Pillar robbing after mining if complete will yield an additional tonnage and increase the total extraction.

Explosives

Explosives used for all mining and development operations are 40 and 60 percent Gelignite.

Both electric and fuse blasting are employed. Ordinary safety fuse is capped in underground magazines and issued with a string attached for tying on to the primer cartridge. No. 6 lead-azide detonators are used for all classes of work except shaft sinking, in which no. 8 detonators are used.

Electric delay igniters and instantaneous electric detonators are inserted into primer cartridges at the magazine and issued ready for use. The higher-strength gelignite is used for all primers.

Explosives and made-up fuses or electric delay igniters are transported from the magazine to working faces in canvas "explosive" bags, a separate bag being used for each material.

Steel-sharpening and drill-repairing

The drill-sharpening and repair shop is located close to the man and supply shaft, with tracks running along outside the building for loading and unloading steel trucks.

Drill steel is delivered to a sorting table, from which it is removed to 3-wheeled trucks for distribution in the shop. The shop has a concrete floor throughout the sharpening section and the loaded trucks can be moved easily to any position required.

Drills are upset and punched on standard sharpeners; they are gaged, and wings are dressed on hot millers. Shanking is done on sharpeners in the usual manner, shank grinders being used to true ends and lugs.

After sharpened steel has cooled, it is reheated for tempering in an oil-furnace, tested on a magnetic indicator, and tempered in water. A separate furnace is used for heating shank ends, which are tempered by quenching them in oil.

Sharpening is done on day-shift only, the capacity of the shop being approximately 2,400 drill steels per day. Equipment consists of:

3 Sullivan-type oil-furnaces (forging heat for bits and shanks).
1 Sullivan-type oil-furnace (tempering heat for bits).

1 Sullivan-type oil-furnace (tempering heat for shanks).
3 No. 50 Leyner sharpeners.
1 Holman double-end hot miller.
1 Morgan hot miller.
2 shank grinders.
1 drill press.
1 power hacksaw.
1 blacksmith's forge.

The drill-repair shop is at one end of the sharpening shop and is equipped to clean, repair, and test machines, hoses, and mountings. No drill repairing is done underground.

Adjacent to the drill shop is a storage shed for equipment such as hoists, motors, and fans. Pipes and fittings are reconditioned and special pipe lengths and manifolds are made up in a portion of the storage shed equipped for the purpose.

Drainage and Pumping

Main pumping stations and sumps are at the main ore shaft on no. 4 and no. 5 haulage levels. Water flows along the levels to the sumps, whence it is pumped to the surface.

In the section between the supply shaft and ore shaft, the porous nature of the country rock and the temperature of the water has made it necessary to convey the water in covered wooden launders. The covering reduces the absorption of water vapor by ventilation currents along the main haulage way.

The maximum volume of water pumped for any considerable period has been 1,500 g.p.m.; the minimum has been 1,000 g.p.m. On no. 4 level the pump capacity was originally 3,100 g.p.m., but on completion of the lower level no. 4 station is used only as a relay station and some of the pump units have been removed to no. 5 level, where the pumping capacity is 2,200 g.p.m.

All pumps are centrifugal type, electrically driven, and discharge into two mains leading to the surface through the main ore shaft.

Pumping cost, including maintenance of pumps and pipe lines, amounted to $53,600 per year, or $0.076 per ton of ore mined from June 1935 to June 1936.

Ventilation

Natural ventilation through shafts and extension of mine workings to surface glory holes is assisted by two main exhaust fans; one is at the north end of the Black Star workings and the other at the south end of the Rio Grande section. Figure 4 shows the position of these fan-shafts in relation to main workings.

The Black Star fan-shaft is equipped with a fan capable of exhausting 100,000 c.f.m. at a pressure not exceeding 3-inch static water gage. The man and supply and Davidson shafts serve as downcast entries in addition to numerous stope openings into the glory holes. Fresh air is circulated thus down through workings to the haulage level and is exhausted through the fan-shaft. In the past this system has provided adequate ventilation.

The Rio Grande workings are ventilated similarly, but, being less extensive, are controlled better. The fan on the south shaft exhausts approximately 40,000 c.f.m. at 3-inch static water gage, which maintains good conditions in this section of the mine during summer or winter.

The chief obstacle to be overcome in the maintenance of proper underground air temperatures is the extremely high surface temperatures that prevail for several months of the year. Temperatures of 100° F. and over are common, and the effect of this hot air in the mine is to preclude the maintenance of ideal temperatures unless an extremely high air velocity through main airways can be obtained.

Auxiliary ventilation in long drifts or crosscuts is provided by 4,000 c.f.m. blowers discharging through 16-inch-diameter galvanized ventpipe to within a short distance of the face. These blowers develop 12-inch static water-gage pressure and under most conditions are satisfactory for distances of 500 feet. For greater distances a booster fan of the same type is used.

Fire Hazards

The small extent of timbered workings, the noncombustible nature of the ore, unless accumulated in large, broken, ore reserves, which is prohibited, and the absence of any combustible gases make the mine almost free from fire hazard. Water hydrants and fire hose long enough to reach to the end of timbered sections are available on each level of all shafts. All electrical substations, hoist rooms, and pump stations are equipped with Foamite extinguishers.

Fire-fighting and rescue teams have not been organized, but forming and active training of such crews is contemplated; they will be maintained for possible emergency.

Safety Methods

A safety committee composed of departmental superintendents, with the general manager as chairman, unifies safety work for all sections of the mine and plant. Each superintendent heads the safety committee in the department he controls. The mine committee is composed of all foremen, shift bosses, trammer bosses and mechanical and electrical foremen. Meetings are held once a month for the purpose of (1) reviewing reports and details of every accident that has occurred since the previous meeting, (2) discussing operating practices and suggestions for rules to prevent similar accidents, (3) reporting places or work that involve or may develop accident hazard, with suggestions as to the best way to eliminate hazards, (4) commenting on equipment, safety clothing, goggles, and other allied protective paraphernalia, and (5) adopting operating safety rules.

The chief problem is to obtain the cooperation of all employees, and educate them to work safely. Individual carelessness, particularly among new men, unfamiliar with underground work, is the cause of most of the injuries suffered. Such accidents cannot be avoided until employees are educated to foresee the possibility of being injured in performing their work and until they willingly cooperate in refraining from slack, imprudent, and reckless methods and short cuts that invariably lead to injury, varying in degree from slight to fatal. Constant explanations, illustrations, warnings, and, when necessary, disciplinary measures are employed with a view of selling safety as a habit to underground workers.

First-aid organization and training

Qualified first-aid men are on duty at the mine casualty station during all working periods. Their duties include (1) application of first-aid dressings and treatments, (2) taking charge of stretcher cases in the mine prior to the arrival of the doctor and assisting in their removal to the surface, (3) maintenance of all stretcher lockers throughout the mine, (4) making reports on all injuries, and (5) issuing protective devices, such as goggles and respirators.

Classes in first-aid training are given by the company doctor at regular periods, under the auspices of the Mount Isa Mines Ambulance Corps, patterned after and affiliated with the St. John Ambulance Association. Such classes are free to employees, and all men are encouraged to become proficient in this valuable work. Shift bosses are required to qualify for a first-aid certificate within a reasonable length of time.

A competition and demonstration is held annually for teams representing different sections of the mine and plant, the reward for the winning team being a replica of the large first-aid trophy for each member of the team. The permanent trophy was purchased jointly by the ambulance corps members and the company.

Wage, contract, and bonus systems

A system of contract payments is used wherever possible in the mine. Approximately 95 percent of underground employees are working on a basis that makes it possible for them to earn more than a base wage. The contract system is influenced to some degree by Industrial Award conditions, precedent, and local prejudice. Although these conditions give rise to undesirable features, the contract system has been found necessary in order to induce performances demanded by modern mining practice.

Contracts

Development, stoping, timbering, and other work that is subject to measurement is done on contract. The system used includes the following principles:

1. Men working on contract must be paid, as a minimum, the prevailing Industrial Award rates applicable to class of work performed.

2. All contracts are paid fortnightly, measurements being made by the mine engineering staff.

3. Contract earnings are divided pro rata among those participating, regardless of individual rating as miner, mucker, etc.

4. Explosives are sold to contractors at landed cost to the company.

5. Prices or rates are determined by the mine superintendent in conjunction with the assistant superintendent and the mine foremen.

6. Changes in rates are made only when marked variations occur in conditions; they are made effective from the beginning of a pay period, with notification given to those concerned.

Contracts for development work, including stope-preparation headings, are based on a set rate per foot of advance. Prices for such work are generally standard throughout the mine, allowance being made for unusual working conditions or other features affecting performance.

Stoping contracts are based on cubic measurement or on tons extracted, whichever is applicable. For such work as stope undercuts, cut-off stopes, or sublevel benching, where ore broken is not currently drawn, the number of cubic yards broken or excavated is the basis for payment.

Payment on a tonnage basis was applied more frequently during the period of carbonate ore mining than at present. Ore mined in glory holes and in filled stopes was paid for on a tonnage basis. Weights for such contracts are determined from the number of cars of ore drawn multiplied by the tonnage factor per car. Adjustment of all tonnage is made to check with the mill weightometer records over weekly periods.

Timbering contracts include such work as erecting chutes, grizzlies, ladderways and tool-cradle slides, and other miscellaneous work. A price is set to cover all work required to complete the particular job. Prices for erecting structures such as chutes or grizzlies, the design for which is standardized, are the same throughout the mine.

Bonus

Ore-transport and drill-sharpening contracts differ in basic principle from those described above and more closely resemble a bonus system.

Ore-transportation crews are paid at a flat rate per ton of ore hauled and hoisted. From the total earnings, wages at the regular scale are deducted, and the balance is divided equally as bonus per shift worked. The tonnage paid for is as recorded by the mill weightometer converted to dry tons. Other haulage chores, such as transporting steel, timber, explosives, and other supplies, are included as regular duties of haulage crews.

A bonus system for drill sharpeners consists of the regular wage-scale payment for a fixed standard, plus a bonus rate for work done in excess of the standard. Men who handle steel, from its entrance to the shop on the sorting table to its exit from the tempering tanks, are included in the bonus.

Wages

Wage rates are fixed according to Industrial Awards and are varied by the State Industrial Arbitration Court. Rates that prevailed in June 1936 for a working period of 88 hours per fortnight are as follows:

Classification	Rate per 8-hour shift
First machine miner (drill runner) ...	$3.76
Second machine miner (drill helper) ..	3.46
Mucker, trucker, or tool nipper	3.36
Timbermen	3.76
Powderman	3.76
Locomotive driver	3.76
Locomotive brakeman	3.45
Electrician	4.83
Pipefitter	3.76
Platman (cage-tender)	3.46
Hoisting-engine driver	4.36
Pumpman	4.40

Costs and Performance

Table 1 contains current operating costs for mining 270,063 tons by glory-hole mining during the first 6 months of 1933.

Table 2 contains current stoping and development cost, together with current capital development and development redemption by the sublevel stope mining - center-heading and benching method for the 12 months from June 1, 1935, to June 30, 1936. Ore mined by this method and hoisted was 703,924 tons.

Organization

Figure 25 is the organization chart of the mining department of Mount Isa Mines, Ltd.